
non numero nisi
ex
libris
John Holland Leavell

S. F. Emmons.

See Biographical Notice, p. 643.

TRANSACTIONS

OF THE

AMERICAN INSTITUTE OF MINING ENGINEERS.

VOL. XLII.

CONTAINING THE PAPERS AND DISCUSSIONS OF 1911.

NEW YORK, N. Y.:
PUBLISHED BY THE INSTITUTE,
AT THE OFFICE OF THE SECRETARY.
1912.

PREFACE.

THIS volume contains all the proceedings, papers, and discussions of the Institute published during 1911, with the following exceptions:

1. Brief obituary notices of members and associates reported as deceased during the year 1911; library accessions and requirements; notices of meetings of the Institute and of other societies; lists of proposed members and associates; changes of address of members; and other announcements of general but temporary interest, furnished to members in *Bulletin* Nos. 49 to 60, during the year 1911.

2. Account of the excursions and entertainments connected with the Wilkes-Barre meeting, June, 1911,[1] and with the San Francisco meeting, October, 1911.[2]

3. The following papers, presented at the San Francisco meeting, which on account of lack of space are carried over to Volume XLIII.:

The Mining Industry of Japan, by Keijiro Nishio, Tokyo, Japan.[3]

The Black Mountain Coal-District, Kentucky, by J. B. Dilworth, Philadelphia, Pa.[4]

The Geology of the Tonopah Mining-District, by Augustus Locke, Goldfield, Nev.[5]

A Modification of the "Gay Lussac" Method for Silver-Bullion Containing Tin, by Luis Emlynn Salas, New York, N. Y.[6]

Notes on the Laramie Tunnel, by D. W. Brunton, Denver, Colo.[7]

The Laws of Igneous Emanation Pressure, by Blamey Stevens, New York, N. Y.[8]

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[1] *Bulletin* No. 55, July, 1911, pp. 584 to 594.
[2] *Idem*, No. 59, November, 1911, pp. xii. to xxxviii.
[3] *Idem*, No. 61, January, 1912, pp. 103 to 147.
[4] *Idem*, No. 62, February, 1912, pp. 149 to 176.
[5] *Idem*, No. 62, February, 1912, pp. 217 to 226.
[6] *Idem*, No. 63, March, 1912, pp. 267 to 278.
[7] *Idem*, No. 64, April, 1912, pp. 357 to 376.
[8] *Idem*, No. 64, April, 1912, pp. 411 to 427.

Physical Data of Igneous Emanation, by Blamey Stevens, New York, N. Y.[9]

The Bearing of the Theories of the Origin of Magnetic Iron-Ores on Their Possible Extent, by Frank L. Nason, West Haven, Conn.

Gold-Mines in Southern Colombia, by F. Pereira Gamba, Tuquerres, Colombia.

4. A few discussions referring to papers contained in Vol. XLI., which were received early in the year 1911, yet in time to be included in said volume.

The publication of the Year Book, containing a revised List of Members and Associates, heretofore usually issued directly after the close of the calendar year, was postponed until after the Annual Business Meeting in February, 1912, in order to have the period covered correspond to the official year of the Institute.

On the other hand, this volume includes the following paper presented at the Canal Zone meeting, which was omitted from Vol. XLI. on account of lack of space :

The Agency of Manganese in the Superficial Alteration and Secondary Enrichment of Gold-Deposits in the United States, by William H. Emmons, Chicago, Ill.

JOSEPH STRUTHERS,
Secretary and Editor.

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[9] *Bulletin* No. 64, April, 1912, pp. 429 to 438.

CONTENTS.

PROCEEDINGS.

PAPERS.

DISCUSSIONS.

OFFICERS.

For the year ending February, 1912.

COUNCIL.*

PRESIDENT OF THE COUNCIL.

CHARLES KIRCHHOFF.................................. ....NEW YORK, N. Y.

(Term expires February, 1912.)

VICE-PRESIDENTS OF THE COUNCIL.

BENJAMIN B LAWRENCE..... ....NEW YORK, N. Y.
JOSEPH W. RICHARDS.......................................SOUTH BETHLEHEM, PA.
ALBERT SAUVEUR.. ...CAMBRIDGE, MASS.

(Term expires February, 1912)

S. B. CHRISTY............................ ....BERKELEY, CAL.
W. A. LATHROP...PHILADELPHIA, PA.
GARDNER F. WILLIAMS................. ....WASHINGTON, D. C.

(Term expires February, 1913.)

COUNCILORS.

KARL EILERS..NEW YORK, N. Y.
ALEX. C. HUMPHREYS...NEW YORK, N. Y.
W. G. MILLER.................... ....TORONTO, CANADA.

(Term expires February, 1912.)

ROBERT E. JENNINGS...NEW YORK, N. Y.
WILLIAM KELLY............................ ....VULCAN, MICH.
CHARLES F. RAND.......... .. ..NEW YORK, N. Y.

(Term expires February, 1913.)

A. E. CARLTON..CRIPPLE CREEK, COLO.
W. J. OLCOTT..................................:..............................DULUTH, MINN.
E. L. YOUNG..NEW YORK, N. Y.

(Term expires February, 1914.)

SECRETARY OF THE COUNCIL AND EDITOR.

†JOSEPH STRUTHERS, 29 W. 39th St..............................NEW YORK, N. Y.

(Term expires February, 1912.)

SECRETARY EMERITUS OF THE COUNCIL.

R. W. RAYMOND................... ....NEW YORK, N. Y.

CORPORATION.

JAMES GAYLEY, President; JAMES DOUGLAS, Vice-President;
FRANK LYMAN, Treasurer;
†JOSEPH STRUTHERS, Secretary and Assistant Treasurer.

DIRECTORS.

THEODORE DWIGHT, ‡ARTHUR L. WALKER, †JOSEPH STRUTHERS.

(Term expires February, 1912.)

JAMES GAYLEY, CHARLES KIRCHHOFF, FRANK LYMAN.

(Term expires February, 1913.)

JAMES DOUGLAS, JAMES F. KEMP, ALBERT R. LEDOUX.

(Term expires February, 1914.)

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* SECRETARY'S NOTE.—The Council is the professional body, having charge of the election of members, the holding of meetings (except business meetings), and the publication of papers, proceedings, etc. The Board of Directors is the body legally responsible for the business management of the Corporation, and is therefore, for convenience, composed of members residing in New York.

† Succeeding R. W. Raymond, resigned Mar. 31, 1911.

‡ Succeeding Charles H. Snow, resigned Apr. 28, 1911.

OFFICERS ELECTED AT ANNUAL MEETING, FEB. 20, 1912.

The list of officers on the preceding page is for the year 1911, the period covered by the contents of this volume of the *Transactions.* But the result of the election at the Annual Business Meeting, February, 1912, although strictly belonging to the next volume, is here published for the convenience of members.

The following officers were elected by vote of the members and associates in person or by proxy at the Annual Meeting, Feb. 20, 1912:

COUNCIL.

President of the Council.

James F. Kemp, ... New York, N. Y.

(To serve for one year Term expires February, 1913.)

Vice-Presidents of the Council.

Karl Eilers, ... New York, N. Y.
Waldemar Lindgren, ... Washington, D. C.
Benjamin B. Thayer, ... New York, N. Y.

(To serve for two years Term expires February, 1914)

Councilors.

John H. Janeway, Jr., ... New York, N. Y.
Sidney J. Jennings, ... New York, N. Y.
Joseph W. Richards, ... So. Bethlehem, Pa.

(To serve for three years. Term expires February, 1915.)

Secretary of the Council and Editor.

Joseph Struthers, ... New York, N. Y.

(To serve for one year. Term expires February, 1913.)

DIRECTORS OF THE CORPORATION.

Edmund B. Kirby, Charles F. Rand, George C. Stone.

To serve for three years. Term expires February, 1915.)

The following are the officers of the Corporation for the year ending February, 1913:

President, James F. Kemp, New York, N. Y.
Vice-President, Edmund B. Kirby, St. Louis, Mo.
Secretary, George C. Stone, New York, N. Y.
Treasurer, Frank Lyman, New York, N. Y.
Assistant Secretary and Assistant Treasurer, Joseph Struthers, New York, N. Y.

PAST OFFICERS.

Presidents.

*David Thomas 1871
R. W. Raymond 1872–1874
*A. L. Holley 1875
*Abram S. Hewitt 1876
*T. Sterry Hunt 1877
*Eckley B. Coxe 1878–1879
*William P. Shinn 1880
*William Metcalf 1881
*Richard P. Rothwell 1882
Robert W. Hunt 1883
James C. Bayles 1884–1885
Robert H. Richards 1886
*Thomas Egleston 1887
William B. Potter 1888
Richard Pearce 1889
*Abram S. Hewitt 1890
John Birkinbine 1891–1892
H. M. Howe 1893
John Fritz 1894
*J. D. Weeks 1895
E. G. Spilsbury 1896
*Thomas M. Drown 1897
Charles Kirchhoff 1898
James Douglas 1899–1900
E. E. Olcott 1901–1902
Albert R. Ledoux 1903–1904
James Gayley 1905
Robert W. Hunt 1906
John Hays Hammond 1907–1908
D. W. Brunton 1909–1910
Charles Kirchhoff 1911
James Gayley (Corporation) 1905–1911

Secretaries.

*Martin Coryell 1871–1872
*Thomas M. Drown 1873–1884
R. W. Raymond 1884–1911
Joseph Struthers .... 1911 ——

Treasurers.

J. Pryor Williamson 1871–1872
*Theodore D. Rand 1872–1903
Frank Lyman .... 1903 ——

* Deceased.

HONORARY MEMBERS.

Year of Election		
1876.	Prof. Richard Åkerman	Stockholm, Sweden.
1909.	Prof. Richard Beck	Freiberg, Germany.
1905.	Andrew Carnegie	New York, N. Y.
1906.	Dr. James Douglas	New York, N. Y.
1888.	Prof. Haton de la Goupillière	Paris, France.
1906.	Sir Robert A. Hadfield	London, England.
1888.	Prof. Hans Hoefer	Leoben, Austria.
1905.	Prof. Henri Louis Le Chatelier	Paris, France.
1899	M. Floris Osmond	Saint Leu, France.
1909.	Alexandre Pourcel	Paris, France.
1911.	Dr. Rossiter W. Raymond	New York, N. Y.
1911.	Prof. Robert H. Richards	Boston, Mass.
1909.	Dr. Ing. H. C. Emil Schroedter	Düsseldorf, Germany.
1906.	John E. Stead	Middlesbrough, England.
1909.	James M. Swank (Associate)	Philadelphia, Pa.
1902.	Prof. Dimitry Constantin Tschernoff	St. Petersburg, Russia.
1910.	Prof. Tsunashiro Wada	Tokyo, Japan.
1907.	Charles D. Walcott	Washington, D. C.

Honorary Members (*Deceased*).

		Year of Decease.
1872.	Bell, Sir Lowthian	1904
1892.	Castillo, A. Del	1895
1902.	Contreras, Manuel Maria	1902
1888.	Daubrée, A.	1896
1884.	Drown, Thomas M.	1904
1890.	Gaetzschmann, Moritz	1895
1873.	Gruner, L.	1883
1891.	Kerl, Bruno	1905
1895.	Le Conte, Joseph	1901
1891.	Lesley, J. P.	1896
1890.	Patera, Adolph	1890
1886.	Percy, John	1889
1888.	Posepny, Franz	1895
1884.	Richter, Theodor	1898
1899.	Roberts-Austen, W. C.	1902
1890.	Serlo, Albert	1898
1880.	Siemens, C. William	1883
1872.	Thomas, David	1882
1873.	Tunner, Peter R. von	1897
1885.	Wedding, Hermann	1908

LIST OF THE MEETINGS OF THE INSTITUTE AND THEIR LOCALITIES FROM ITS ORGANIZATION TO OCTOBER, 1911.

No.	Place.	Date.	Trans. Vol.	Trans. Page.
1	Wilkes-Barre, Pa *	May, '71	1	3
2.	Bethlehem, Pa.	Aug., '71.	1	10
3	Troy, N Y	Nov., '71.	1	13
4.	Philadelphia, Pa	Feb., '72	1	17
5.	New York, N. Y.*	May, '72	1	20
6.	Pittsburg, Pa	Oct, '72	1	25
7.	Boston, Mass	Feb., '73	1	28
8.	Philadelphia, Pa *	May, '73	2	3
9	Easton, Pa	Oct., '73	2	7
10.	New York, N. Y	Feb, '74	2	11
11	St Louis, Mo.*	May, '74	3	3
12.	Hazleton, Pa	Oct, '74	3	8
13	New Haven, Conn	Feb., '75	3	15
14.	Dover, N. J.*	May, '75	4	3
15.	Cleveland, O	Oct., '75	4	9
16	Washington, D. C.	Feb, '76	4	18
17.	Philadelphia, Pa.†	June, '76	5	3
18.	Philadelphia, Pa	Oct., '76	5	19
19.	New York, N Y.	Feb, '77	5	27
20.	Wilkes-Barre, Pa.*	May, '77	6	3
21.	Amenia, N. Y	Oct, '77	6	10
22.	Philadelphia, Pa	Feb., '78	6	18
23.	Chattanooga, Tenn.*	May, '78	7	3
24.	Lake George, N. Y.	Oct, '78	7	103
25.	Baltimore, Md *	Feb, '79	7	217
26	Pittsburg, Pa	May, '79	8	3
27.	Montreal, Canada	Sept., '79	8	121
28.	New York, N Y.*	Feb, '80	8	275
29.	Lake Superior, Mich	Aug, '80	9	1
30.	Philadelphia, Pa *	Feb., '81	9	275
31	Staunton, Va.	May, '81	10	1
32.	Harrisburg, Pa	Oct., '81	10	119
33	Washington, D. C.*	Feb., '82	10	225
34.	Denver, Colo	Aug., '82	11	1
35	Boston, Mass.*	Feb, '83	11	217
36.	Roanoke, Va	June, '83	12	3
37.	Troy, N Y	Oct., '83	12	175
38.	Cincinnati, O.*	Feb, '84	12	447
39.	Chicago, Ill	May, '84	13	1
40.	Philadelphia, Pa	Sept., '84	13	285
41.	New York, N. Y.*	Feb., '85	13	585
42.	Chattanooga, Tenn.	May, '85	14	1
43.	Halifax, N. S	Sept., '85	14	307
44.	Pittsburg, Pa.*	Feb., '86	14	587
45.	Bethlehem, Pa	May, '86	15	lxiii.
46.	St Louis, Mo.	Oct., '86	15	lxx.
47.	Scranton, Pa.*	Feb, '87	15	lxxvii.
48.	Utah and Montana	July, '87	16	xvii.
49.	Duluth, Minn	July, '87	16	xxiv.
50.	Boston, Mass.*	Feb., '88	16	xxviii.
51.	Birmingham, Ala	May, '88	17	xix.
52.	Buffalo, N Y.	Oct., '88	17	xxiv.
53.	New York, N. Y *	Feb, '89	17	xxxi.
54.	Colorado	June, '89	18	xvii.
55.	Ottawa, Canada	Oct, '89	18	xxiv.
56.	Washington, D. C *	Feb, '90	18	xxx.
57.	New York, N Y	Sept, '90	19	vii.
58.	New York, N Y.*	Feb, '91	19	xxv.
59.	Cleveland, O	June, '91	20	xvi.
60.	Glen Summit, Pa	Oct., '91	20	lxi.
61.	Baltimore, Md.*	Feb., '92	21	xix.
62.	Plattsburgh, N. Y.	June, '92	21	xxxiii.
63.	Reading, Pa	Oct., '92	21	xliv.
64.	Montreal, Canada*	Feb, '93	21	lii.
65.	Chicago, Ill	Aug., '93	22	xiii.
66.	Virginia Beach, Va.*	Feb, '94	24	xvii.
67.	Bridgeport, Conn	Oct., '94	24	xxxv.
68	Florida ‡	Mar., '95	25	xix.
69.	Atlanta, Ga	Oct., '95	25	xxxiii.
70.	Pittsburg, Pa *	Feb, '96	26	xvii.
71.	Colorado	Sept, '96	26	xxix.
72.	Chicago, Ill	Feb., '97	27	xvii.
73	Lake Superior	July, '97	27	xxx.
74.	Atlantic City, N J.*	Feb, '98	28	xvii.
75.	Buffalo, N Y.	Oct., '98	28	xxxvi.
76.	New York, N. Y.*	Feb, '99	29	xvii.
77.	California	Sept., '99	29	xlix.
78.	Washington, D. C.*	Feb., '00	30	xix
79.	Canada	Aug., '00	30	xlv.
80.	Richmond, Va *	Feb., '01	31	xix.
81	Mexico	Nov., '01	32	cxviii.
82.	Philadelphia, Pa. §	May, '02	33	xxxv.
83.	New Haven, Conn.	Oct., '02	33	xlvii.
84	Albany, N. Y.*	Feb., '03	34	xxiii.
85	New York, N. Y	Oct, '03	34	lxi.
86.	Atlantic City, N. J.*	Feb., '04	35	xxiii.
87.	Lake Superior	Sept., '04	35	xliii.
88.	Washington, D. C	May, '05	36	xlii.
89.	British Columbia	July, '05	36	liii.
90.	Bethlehem, Pa	Feb., '06	37	xli.
91.	London, England	July, '06	37	xlviii.
92.	New York, N. Y	April, '07	38	lii.
93.	Toronto, Canada	July, '07	38	lix.
94	New York, N. Y	Feb., '08	39	xli.
95.	Chattanooga, Tenn	Oct., '08	39	xlviii.
96	New Haven, Conn	Feb, '09	40	xli.
97.	Spokane, Wash	Sept., '09	40	xlviii.
98.	Pittsburg, Pa	Mar., '10	41	xxxviii.
99.	Canal Zone	Nov., '10	41	xlv.
100	Wilkes-Barre, Pa	June, '11	42	xxxiv.
101.	San Francisco, Cal	Oct., '11	42	xliv.

* Annual meeting for the election of officers. The rules were amended at the Chattanooga meeting, May, 1878, changing the annual election from May to February.

† Begun in May at Easton, Pa., for the election of officers, and adjourned to Philadelphia.

‡ Begun in February at New York City, for the election of officers, and adjourned to Florida.

§ " " " " " " " " " " " " " to Philadelphia.

PUBLICATIONS.

The publications of the Institute comprise:

Transactions.

The volumes of *Transactions*, which are published annually, contain the list of officers, rules, etc., the Proceedings, and the papers revised for final publication. These single volumes are for sale as follows, in paper covers:

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"*The Genesis of Ore-Deposits*," comprising the famous treatise of the late Professor Franz Pošepný, with the successive discussions thereof by Le Conte, Blake, Winchell, Church, Emmons, Becker, Cazin, Rickard, and Raymond; also, later papers by Van Hise, Emmons, Weed, Lindgren, Vogt, Kemp, Blake, Rickard, and others, and the discussions of these papers by De Launay, Beck, and many others; also a complete bibliography of Institute papers and discussions on this subject from 1871 to 1902. 825 pages.

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1. The Minutes of the Proceedings of each meeting.

2. Such of the papers presented or read by title at each meeting as are furnished by the authors and approved by the Council for full publication. These papers are published separately in pamphlet form, and are marked "subject to revision." Beyond the *Bulletin* edition, a small supply is retained to meet subsequent demand. The stock is nearly complete from 1880. These papers are for sale at the following prices:

No. of Pages.	Single Copies.	10 Copies.	20 Copies.
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161 to 176	0.60	4.50	7.00

Authors' Edition of Pamphlets.

Extra copies of pamphlets, if ordered before the printing of the *Bulletin*, will be furnished to members of the Institute at special rates.

CONSTITUTION.

[ADOPTED FEB. 21, 1905.]

ARTICLE I.

NAME AND OBJECT.

SEC. 1 This Institute is incorporated under the Membership Corporation Law of the State of New York; its corporate name is AMERICAN INSTITUTE OF MINING ENGINEERS; and its objects are such as are stated in its Certificate of Incorporation.

ARTICLE II.

MEMBERS.

SEC. 1. The membership of the Institute shall comprise four classes, namely: (1) Members; (2) Honorary Members; (3) Associates; and (4) Honorary Associates. Only Members and Associates residing within the United States of America, Republic of Mexico and Dominion of Canada shall be entitled to vote at the meetings of the Institute.

SEC. 2. All Members, Honorary Members, Associates and Honorary Associates of the American Institute of Mining Engineers as the same existed on the day of the incorporation of this Institute, are Members, Honorary Members, Associates and Honorary Associates, respectively, of this Corporation.

SEC. 3. The following classes of persons shall be eligible for membership in the Institute, namely: as Members and Honorary Members, all professional mining engineers, geologists, metallurgists or chemists, and all persons practically engaged in mining, metallurgy or metallurgical engineering; as Associates and Honorary Associates, all persons desirous of being connected with the Institute who, in the opinion of the Council, are suitable.

SEC. 4. Every candidate for election as a Member or Associate of the Institute must be proposed for election by at least three Members or Associates; must be approved by the Committee on Membership, as prescribed in the By-Laws; and must be elected by the Council. Not less than three-fourths of the votes cast shall be necessary to an election. Every person so elected shall become a Member or Associate, as the case may be, upon payment of his first dues as hereinafter prescribed. Each candidate for Honorary Member or Honorary Associate, must be recommended by at least ten Members or Associates; must be approved by the Council; and must be elected by ballot at a meeting of the Board of Directors by the unanimous vote of all the Directors present; provided, however, that the number of Honorary Members and Honorary Associates shall not at any time exceed twenty.

SEC. 5. If any person elected a Member or Associate does not, within sixty days after notice of his election, accept the same and pay his initiation fee and dues for the current year, his election may be cancelled at the discretion of the Council.

SEC. 6. The Council may at any time change the classification of a person elected as an Associate so as to make him a Member, or vice versa. All Members and Associates shall be equally entitled to the privileges of membership, provided that Honorary Members, Honorary Associates, and Members and Associates whose Post-Office addresses shall be outside of the United States, Mexico and Canada, shall not be entitled to vote.

ARTICLE III.

DUES.

SEC. 1. The dues of Members and Associates shall be Ten Dollars per annum, payable in advance on the first day of each Calendar year. Each newly elected Member or Associate shall pay, when notified of election, an initiation fee of Ten Dollars in addition to the dues for the current year. Honorary Members and Honorary Associates shall not be liable to initiation fee or dues. Any Member or Associate in arrears for one year may, at the discretion of the Council, be deprived of the receipt of publications or stricken from the list of Members, provided that he may be restored to membership by the Council on payment of all arrears or may be again proposed and elected after an interval of three years.

SEC. 2. Any Member or Associate not in arrears may become, by the payment of One Hundred and Fifty Dollars at one time, a Life Member or Associate; and shall not be liable thereafter to annual dues.

ARTICLE IV.

BUSINESS MEETINGS OF THE INSTITUTE.

SEC. 1. The annual meeting of the Institute for the election of Directors and transaction of other business shall take place on the third Tuesday in February in each year. A report of the financial condition of the Institute and an abstract of the accounts shall be furnished by the Directors, and presented at each annual meeting.

SEC. 2. Special business meetings of the Institute may be held at such times and places as the Board of Directors may appoint, upon notice to all Members and Associates entitled to vote, directed to each at his last known Post-Office address, and mailed in the City of New York not less than twenty days before the date fixed for such meeting.

SEC. 3. At all business meetings of the Institute the presence of nine Members and Associates shall constitute a quorum.

SEC. 4. At all business meetings of the Institute Members and Associates may vote either in person or by proxy, but no Member or Associate in arrears since the last annual meeting shall be entitled to vote.

ARTICLE V.

OTHER MEETINGS OF THE INSTITUTE.

SEC. 1. All meetings of the Institute other than business meetings shall be held at such times and places as the Council may appoint. Notice of all such meetings shall be given to all Members and Associates by mail.

ARTICLE VI.

Directors and Officers.

Sec. 1. The business and financial affairs of the Institute shall be managed by a Board of Directors, who shall be elected at the annual meeting in the manner prescribed in the Certificate of Incorporation.

Sec. 2. The officers of the corporation shall be a President, Vice-President, Secretary and Treasurer, who shall be elected by the Directors from among their number. All such officers shall be elected at the first meeting of the Board of Directors after each annual meeting of the corporation, and shall hold office for one year or until their successors are elected and qualify.

The duties of all officers shall be such as usually pertain to their offices, respectively, together with such other duties as may from time to time be prescribed for them by the By-Laws. The Treasurer shall give a bond for the faithful performance of his duties in a sum to be fixed by the Board of Directors, but at the expense of the Institute.

Sec. 3. In the event of a vacancy occurring in the Board of Directors by death, resignation or otherwise, the remaining members of the Board may, by a majority vote, elect a successor to fill the vacancy, who shall continue in office until the next annual meeting or until his successor shall have been chosen.

Sec. 4. The Board of Directors may, in its discretion, declare the place of any Director vacant, on his failure for any reason, to attend three successive meetings of the Board. Any Director who shall under this section or in any other manner cease to be a member of the Board shall, at the same time, be held to have vacated any other office to which he shall previously have been elected; and the Board shall elect a new incumbent to the said vacant office.

Sec. 5. The Board of Directors may from time to time appoint from their own number standing and special committees, and may delegate to such committees such duties as they may see fit.

ARTICLE VII.

Meetings of the Board of Directors.

Sec. 1. A regular meeting of the Board of Directors for the election of officers and the transaction of other business shall be held on the third Tuesday in February in each year, after the adjournment of the annual meeting of the Institute.

Sec. 2. Special meetings of the Board of Directors, at which any business may be transacted, may be called to meet at any time at the office of the Institute in the City of New York, by notice in writing mailed at least five days before the meeting, by the Secretary to each member of the Board at his last known Post-Office address, signed either by the President or the Vice-President or by three members of the Board.

Sec. 3. At all meetings of the Board of Directors the presence of five members shall constitute a quorum.

ARTICLE VIII.

The Council.

Sec. 1. The professional, technical, scientific and social interests of the Institute shall be committed to the supervision of a Council composed of a President

of the Council, six Vice-Presidents of the Council, a Secretary of the Council and nine Councilors, who shall be elected from among the Members and Associates of the Institute in the manner hereinafter prescribed Members of the Council may or may not be members of the Board of Directors.

SEC. 2. The President of the Council shall be elected for one year, and no person shall be eligible for immediate re-election to this office who shall have held the same for two consecutive years.

After the first year Vice-Presidents of the Council shall be elected to serve for two years, and Councilors shall be elected to serve for three years. No Vice-President of the Council or Councilor shall be eligible for immediate re-election to the same office at the expiration of the term for which he was elected. The Secretary of the Council shall be elected annually.

SEC. 3. At the first annual meeting, to be held in the year 1905, there shall be elected a President of the Council to serve for one year, a Secretary of the Council to serve for one year, three Vice-Presidents of the Council to serve for one year, three Vice-Presidents of the Council to serve for two years, three Councilors to serve for one year, three Councilors to serve for two years, and three Councilors to serve for three years. At each subsequent annual meeting there shall be elected a President of the Council to serve for one year; a Secretary of the Council to serve for one year; three Vice-Presidents of the Council to serve for two years; and three Councilors to serve for three years. The term of office of all Members of the Council shall continue until the adjournment of the meeting at which their successors are elected.

SEC. 4. Vacancies in the Council may occur by death or resignation; or the Council may, by the vote of a majority of all its members, declare the place of any officer or member of the Council vacant, on his failure for one year, from inability or otherwise, to attend the regular meetings or perform the duties of his office. All vacancies shall be filled by the appointment of the Council, and any person so appointed shall hold office for the remainder of the term for which his predecessor was elected or appointed; *provided* that the said appointment shall not render such person ineligible for election to the Council at the next meeting.

SEC. 5. The presence of five members of the Council shall constitute a quorum; but the Council may appoint an Executive Committee, or any business coming within the authority of the Council may be transacted at a regularly-called meeting thereof, at which less than a quorum may be present, subject to the approval of a majority of the Council subsequently given in writing to the Secretary and recorded by him with the minutes.

SEC. 6. The election of the Council shall take place at the regular annual meeting of the Institute. Nominations for members of the Council may be sent in writing to the Secretary accompanied with the names of the proposers at any time not less than thirty days before the annual meeting; and the Secretary shall, not less than two weeks before said meeting, mail to every Member or Associate entitled to vote a list of all nominations for each office so received, together with the names of the persons ineligible for election to each office; and if the Council or a Committee thereof, appointed for the purpose, shall have recommended any nomination, such recommendation may also be sent to the Members and Associates with the list of all nominations made.

ARTICLE IX.

Meetings of the Council.

Sec. 1. Meetings of the Council shall be held at such times and places as the President of the Council or one of the Vice-Presidents of the Council may appoint.

Sec. 2. A meeting of the Council may be held on the day of the annual meeting of the Institute without previous notice. Written notice of all other meetings of the Council, specifying the time and place of such meeting, signed by the Secretary, shall be mailed to every member of the Council at his last known Post-Office address at least ten days before the date of the meeting.

ARTICLE X.

Papers and Publications.

Sec. 1. The Council shall have power to decide as to the acceptance and publication of any professional papers presented to the Institute, subject to such conditions as the Board of Directors may prescribe.

Sec. 2. The copyright of all professional papers communicated to and accepted by the Institute shall be vested in it, unless otherwise expressly agreed between the Council and the author. The Institute shall not assume responsibility for any statements of fact or opinion advanced in the papers or discussions at its meetings. Neither the Council nor the Institute shall officially approve or disapprove any technical or scientific opinion or any proposed enterprise, outside of the management of the meetings, discussions and publications of the Institute, and the conduct of its business affairs by the Board of Directors.

Sec. 3. Special Committees may from time to time be appointed by the Council to make investigations and prepare reports for presentation to the Institute, but no action shall be taken binding the Institute for or against the conclusions embodied in any such reports.

ARTICLE XI.

Suspensions and Expulsions.

Sec. 1. Any member of the Institute who shall be convicted of a crime involving, in the opinion of the Board of Directors, moral turpitude, shall, upon the passage by the Board of Directors of a resolution declaring the crime for which he has been convicted to be of such character, be thereupon dropped from membership in this Institute.

Sec. 2. Any member of the Institute may be suspended or expelled for mis conduct by the Board of Directors, after charges setting forth such misconduct shall have been prepared by the Council and filed in writing with the Board. Upon the receipt of such charges in writing, the Board may, in its discretion, suspend such member pending a hearing and determination thereupon. As soon as may be after the receipt of such charges, the Board shall fix a date for a hearing thereupon and shall give to the accused member notice thereof in writing, mailed to him at his last known Post-Office address not less than thirty days before said date, accompanied by a full copy of the charges and a copy of the second, third and fourth sections of this article.

Sec. 3. Upon the day fixed for the hearing, the accused member may appear before the Board, either in person or by an accredited representative; hear any

witnesses who may be called in support of the charges and at his option cross-examine the same; and hear read any documentary evidence offered in support of the charges. The accused may, in his discretion, produce and examine witnesses in his defence, and submit documentary evidence, including a statement from himself in writing. After the conclusion of the hearing, the Board of Directors shall consider and vote to approve or disapprove the charges. If the Board shall, by a vote of two-thirds of its members, declare the charges sustained, it may suspend the member for a stated period or expel him.

SEC. 4. If the accused member shall not appear at the hearing, and shall within three months thereafter file with the Board an affidavit stating that he had not received notice of the charges against him in time to enable him to present his defence, the Board shall fix a date for a re-hearing within three months from the receipt of such affidavit and shall immediately notify the accused member by mail of such date. Upon the re-hearing, the accused shall have the same privilege of presenting his defence as he would have had upon the original hearing; and after the defence is presented, the Board shall take a new vote upon the charges, the result of which shall be conclusive.

SEC. 5. All interests in the property of the Institute of persons resigning, or otherwise ceasing to be Members or Associates, shall vest in the Institute.

ARTICLE XII.

AMENDMENTS.

SEC. 1. This Constitution or any Article or Section thereof may be amended at any annual meeting by a two-thirds vote of all the members present in person or by proxy, *provided* that notice of the proposed amendment shall have been given in writing at a previous meeting, and *provided also* that the amendment or amendments so adopted shall have been printed and mailed to all Members and Associates not later than thirty days before the annual meeting. Any amendment or amendments approved by a majority of the votes cast shall be deemed to have been adopted, and shall become a part of this Constitution. The Secretary shall forthwith print and distribute to Members and Associates an announcement of the result of said vote, and if any amendment or amendments shall have been adopted, a copy of the section or sections so amended.

BY-LAWS.

[ADOPTED FEB. 21, 1905. AMENDED FEB. 20, 1906, NOV. 16, 1906, AND JAN. 5, 1909.]

I. PRESIDING OFFICERS.

At all Business meetings of the Institute the President, or, in his absence, the Vice-President, or, in the absence of both of them, any other member of the Board of Directors to be chosen by the meeting, shall preside.

At all other meetings of the Institute the President of the Council or, in his absence, one of the Vice-Presidents, if present, shall preside.

II. ORDER OF BUSINESS.

At each Business meeting of the Institute the order of business shall be as follows:

1. Reading of minutes of preceding meeting.
2. Report of the President.
3. Report of the Treasurer.
4. Report of the Secretary.
5. Election of Directors.
6. Election of Members of the Council.
7. Reports of Standing Committees.
8. Reports of Special Committees.
9. Special Orders.
10. Miscellaneous business.

This order of business may be changed by a vote of a majority of the Members and Associates present in person or by proxy.

The usual parliamentary rules shall govern all meetings of the Institute except in cases otherwise provided by the Constitution or the By-Laws.

At all sessions of the Institute other than business meetings, the order of proceedings and the time of adjournment shall rest in the discretion of the presiding officer.

III. SECRETARY.

The Secretary shall keep a record of the proceedings of all meetings of the Institute. He shall be custodian of the Corporate Seal, of the Minute Books, and of all Legal Documents belonging to the Institute. He shall conduct, on behalf of the Institute, all correspondence relating to business matters, except such as pertains directly to the office of the Treasurer.

He shall notify all officers and Directors and Members of the Council, and all Members of Committees of their election and appointment; shall issue notices of all meetings of the Board, and of the annual and other meetings of the Institute; and shall, in calling special meetings of the Directors, specify the object of such meeting.

IV. SECRETARY OF THE COUNCIL.

The Secretary of the Council shall act as the Clerk of that body at all of its meetings and at all meetings of the Institute called for the discussion of professional, technical or scientific matters, or for any other purpose than the transaction of business.

He shall be custodian of all technical or scientific papers submitted to the In-

stitute for its consideration, shall have charge of the editing and printing of all material published by the Institute, and of the distribution thereof. On the first day of May following the year in which each volume of *Transactions* is printed, he shall turn over to the Library Committee all copies of the same not theretofore distributed by him. He shall have charge of all the correspondence of the Institute relating to other than business affairs.

The Secretary of the Council shall receive a salary to be fixed by the Board of Directors. He may appoint an Assistant with the title of Editor, who shall likewise receive a salary to be fixed by the Board of Directors.

The Secretary of the Council may or may not be the same person as the Secretary of the Institute.

V. Assistant Secretary.

The Secretary may, with the approval of the Board of Directors, appoint an Assistant to whom both he and the Secretary of the Council may delegate such of his or their duties as he or they may see fit. This Assistant Secretary shall receive such salary as shall be fixed by the Board of Directors, which shall cover his services both to the Secretary and to the Secretary of the Council.

VI. Treasurer.

The Treasurer shall collect and, under the direction of the Board of Directors, shall disburse all funds of the Institute. He shall keep regular accounts in books belonging to the Institute, which shall be open to any member of the Board of Directors. He shall report in writing at each annual meeting of the Institute and at every meeting of the Board of Directors at which such report shall be called for, the balance of money on hand, and any existing appropriation which may affect the same.

His accounts shall be audited annually by a Committee of three Members or Associates to be appointed by the President at least thirty days prior to the annual meeting in each year, which Committee shall report thereon at such annual meeting.

The Treasurer may, at his discretion, place funds of the Institute, not at any time exceeding $5,000, in a special account in a Bank or Trust Company, subject to the draft of the Assistant Treasurer, and may delegate to the Assistant Treasurer the duty of paying, out of this account, the current expenses of the Institute.

The Treasurer shall be solely responsible to the Institute for all moneys received, whether the same are entrusted to the Assistant Treasurer or not.

VII. Assistant Treasurer.

The Treasurer may appoint, with the approval of the Board of Directors, an Assistant Treasurer, to whom he may delegate the duty of conducting the correspondence incidental to the office of Treasurer, of receiving and depositing in bank to the credit of the Institute all moneys received, and of paying, out of the special account upon which he may be authorized to draw, the necessary expenses of the Institute. The Treasurer may require of him a bond, running to the Treasurer personally, in an amount not exceeding $5,000, the expense of which shall be borne by the Institute.

The Assistant Treasurer shall receive such compensation as shall be fixed by the Board of Directors.

The offices of the Assistant Secretary and of the Assistant Treasurer may, if

so desired by both the Secretary and the Treasurer and approved by the Board of Directors, be united in the same person, who shall then receive the salary of both offices.

The Assistant Treasurer may, with the approval of the Board of Directors, employ such persons as are necessary to constitute a clerical and office force for himself, the Assistant Secretary and the Secretary of the Council, at such salaries as shall be approved by the Board of Directors. He shall, if the offices of Assistant Secretary and Assistant Treasurer be united in the same person, be the immediate superior of all such employees, unless the Secretary of the Council or the Treasurer be present, in which event either of them shall be the superior of all employees, including their respective assistants.

VIII. Standing Committees.

The Standing Committees of the Institute shall be three in number, known respectively as the Finance Committee, the Library Committee and the Committee on Membership.

The Finance Committee and the Library Committee shall each consist of three members of the Board of Directors, and shall be appointed by the President at the first meeting of the Board, after the annual meeting in each year.

The Committee on Membership shall consist of five Members of the Council, and shall be appointed by the President of the Council, at the first meeting of the Council after the first annual meeting in each year.

IX. Finance Committee.

It shall be the duty of the Finance Committee to inquire into and examine the financial condition of the Institute, and to consider ways and means of increasing its revenues and of limiting its expenses. It shall report from time to time to the Board as often as it may deem expedient, and whenever it shall be directed so to do; and the Treasurer shall at all times furnish it with such statements and information as it may desire.

It shall determine the investment of such surplus moneys as shall from time to time accrue to the Institute. It shall, at least once in each year, examine the securities belonging to the Institute in the custody of the Treasurer, and report thereon to the Board.

It may, at any time, examine the books and vouchers of the Treasurer and Assistant Treasurer.

The Treasurer shall not be a member of the Finance Committee, but shall attend the meetings of the same if requested to do so.

X. Library Committee.

The Library Committee shall be the custodian of all books in the Institute Library and of additions thereto; also of all back numbers of the *Transactions* of the Institute. It shall, on the first day of May, of each year, receive from the Secretary of the Council, and receipt for same to him, all the volumes of *Transactions* for the preceding year, not then distributed by said Secretary.

It shall cause to be kept, under the direction of the Assistant Secretary, a catalogue of all books in the Library and an account in ledger form of all volumes of *Transactions* in its custody, in which shall be charged to it all volumes delivered to it, and in which shall be credited all volumes taken from its custody for sale or for any other purpose.

The receipts from the sale of any volume of *Transactions* taken from the custody of the LIBRARY COMMITTEE shall be credited to the LIBRARY COMMITTEE on the books of the Treasurer, and devoted to the general purposes of the Institute.

XI. COMMITTEE ON MEMBERSHIP.

All nominations for Members or Associates of the Institute shall be submitted to and passed upon by the COMMITTEE ON MEMBERSHIP, who shall report thereon to the Council. It shall receive and consider all communications respecting candidates, and shall make diligent inquiry as to the character and qualifications of each one. Its proceedings shall be secret and confidential.

No member of the Committee shall propose any candidate.

XII. ELECTION OF MEMBERS.

After the COMMITTEE ON MEMBERSHIP shall have reported to the Council its conclusions as to the acceptability of each candidate, the Council shall vote upon the same.

Two negative votes of members of the Council present shall prevent the election of any candidate. No person shall be proposed for election to the Institute within one year after his name shall have been rejected by the Council.

XIII. UNITED ENGINEERING SOCIETY.

The Board of Directors shall, at its first meeting after the adoption of these By-Laws, designate three Members or Associates of this Institute to be representatives of this Institute upon the Board of Trustees of the UNITED ENGINEERING SOCIETY, making at the same time provision for the expiration of the terms of office of said representatives, as provided in the By-Laws of the said UNITED ENGINEERING SOCIETY.

At the last meeting of the Board of Directors prior to the first day of each January thereafter, the Board shall designate a Member or Associate of this Institute to be a representative of this Institute upon the Board of Trustees of the said UNITED ENGINEERING SOCIETY for a period of three years beginning at the next ensuing annual meeting of said Society.

At any time when a vacancy shall occur in the representation of this Institute in the Board of Trustees of said Society, by reason of the death, resignation or removal of any such representative therein, the Board of Directors of this Institute shall designate a Member or Associate to fill such unexpired term.

XIV. PUBLICATIONS.

The publications of the Institute shall include a periodical, called the *Bulletin of the American Institute of Mining Engineers*, which shall contain reports of proceedings, professional papers, notices, and other matter of interest to members. From the annual dues paid by each Member or Associate, five dollars shall be deducted and applied as a subscription to the *Bulletin* for the year covered by such payment.

XV. AMENDMENTS.

These By-Laws may at any time be altered or amended by a vote of two-thirds of the Board of Directors, or by the Members, at a business meeting of the Institute, in the same manner provided for amendments of the Constitution in Article XII. thereof.

ANNUAL MEETING OF THE INSTITUTE.

At the Annual Business Meeting of the Institute, held Feb. 21, 1911, the following officers were elected:

COUNCIL.

President.

(To serve for one year)

CHARLES KIRCHHOFF, ... New York, N. Y.

Vice-Presidents.

(To serve for two years)

S. B. CHRISTY, ... Berkeley, Cal.
W. A. LATHROP, ... Philadelphia, Pa.
GARDNER F. WILLIAMS, ... Washington, D. C.

Secretary.

(To serve for one year.)

R. W. RAYMOND, ... New York, N. Y.

Councilors.

(To serve for three years.)

A. E. CARLTON, ... Cripple Creek, Colo.
W. J. OLCOTT, ... Duluth, Minn.
E. L. YOUNG, ... New York, N. Y.

DIRECTORS.

(To serve for three years)

JAMES DOUGLAS, ... New York, N. Y.
JAMES F. KEMP, ... New York, N. Y.
ALBERT R. LEDOUX, ... New York, N. Y.

[SECRETARY'S NOTE.—The complete list of all officers of the Institute will be found on p. vii. of this volume. The following explanation, first published in *Bi-Monthly Bulletin*, No. 8, March, 1906, p. viii., is here repeated in order to recall to old members, and convey to new ones, the relations of the two governing bodies as determined by the Certificate of Incorporation of the Institute, and the Constitution and By-Laws adopted in accordance therewith.

The body legally responsible for the business management is the Board of nine Directors (three elected annually to serve three years), which elects its own officers. This body, for reasons of practical convenience, is composed of well-known members residing in New York City, and able to attend, without serious inconvenience or expense, the necessary meetings of the Board. The officers of this Board are legally the officers of the Institute. But, apart from business management, the Board exercises no control over the election of members, or the professional and technical work of the Institute, except that its vote is required to elect honorary members, upon the recommendation of the Council.

The Council is a body constituted in all respects (except that it has no Treas-

urer) like the Council existing before the incorporation of the Institute, in January, 1905, and charged with all duties and powers, except those which the Board of Directors must legally perform. It elects members, appoints the times and places of professional meetings, and controls the publication and distribution of papers and volumes, etc. Its members (President, Vice-Presidents and Councilors) are elected by the members of the Institute, voting in person or by proxy, and after publication of the nominations received; and it is intended to represent, as far as practicable, both the professional and the geographical distribution of the membership. Consequently, whatever professional honor attaches to official position belongs to membership in the Council, rather than in the legal Board of Directors. This remark implies no disparagement of the members of the latter body, every one of whom has served, or is now serving, as a member of the Council. But it is only fair to explain that their election and continued re-election as Directors is simply a matter of legal convenience.]

Proposed Amendment to the Constitution.

The proposed amendment of Article III. of the Constitution, changing the annual dues from $10 to $15, notice of which was originally given at the Annual Meeting of February, 1909, and the consideration of which was postponed by the Annual Meeting of February, 1910, was fully discussed, and by unanimous vote of 501 members, present in person and by proxy, it was

Voted.—That the consideration of the proposed amendment to Article III. of the Constitution, whereby the word "fifteen" is substituted for the word "ten" in the first line of said Article, be postponed to an adjourned session of this meeting, to be held at a time fixed by the Board of Directors upon due notice being given as provided by Article IV., Section 2, of the Constitution for the calling of special business meetings.

Provided that, if the Board of Directors shall not call such adjourned session, then the consideration of this proposed amendment shall be in order at the Annual Meeting of February, 1912, and

Provided also that, for the purpose of the said consideration of this amendment, or any proposed amendment thereof, new proxies shall be sent to all members and associates entitled to vote, which shall be so drawn as to cover the questions thus concerned, and permit an intelligent vote thereon.

PROCEEDINGS OF THE BOARD OF DIRECTORS.

The following acts of the Directors are reported for the information of members:

At a meeting held Dec. 9, 1910, Dr. Tsunashiro Wada, of Tokyo, Japan, was, upon the recommendation of the Council, unanimously elected an Honorary Member of the Institute.

At a meeting held Jan. 11, 1911, Mr. Theodore Dwight, Treasurer of the Land Fund Committee, presented a report of the work done by this committee during the year 1910, which shows a total of subscriptions collected during the year of $2,070, and promised subscriptions of $16,000. Payments were made by this committee on the land mortgage during the year amounting to $3,000, reducing the debt of the land fund from $88,000, Jan. 1, 1910, to $85,000, Jan. 1, 1911. The deferred payments, amounting to $16,000, together with the present balance on hand of $209.75, totaling $16,209.75, will further reduce the balance due on the land mortgage to $68,790.25.

Mr. Theodore Dwight was unanimously elected a trustee of the United Engineering Society, to serve for a term of three years, succeeding Mr. Charles Kirchhoff, whose term expired January, 1911.

At a meeting held Feb. 21, 1911, the following officers were elected: *President*, James Gayley; *Vice-President*, James Douglas; *Secretary*, R. W. Raymond; *Treasurer*, Frank Lyman.

The following standing committees were appointed to serve during the ensuing year:

Finance Committee: James Douglas, Theodore Dwight, and Albert R. Ledoux.

Library Committee: James F. Kemp, Charles H. Snow, and R. W. Raymond.

Financial Statement.

The following statement of receipts and expenditures from Jan. 1 to Dec. 31, 1910, is published by authority of the Board of Directors:

RECEIPTS.

Balance from statement of January, 1910,		$5,548.59
Annual dues,*	$34,148.88	
Life memberships,	880.00	
Initiation fees,	1,829 74	
Binding of *Transactions*,	3,373 10	
Sale of publications, electrotypes, advertising, and miscellaneous receipts,	13,856.45	
		54,088.17
Interest on bank deposits,		206.55
		$59,843 31

DISBURSEMENTS.

Printing Vol XL. of the *Transactions*, *Bulletin*, extra pamphlets, and advertising expenses, etc.,	$14,574.16	
Printing circulars and ballots,	196.70	
Binding Vol. XL. of the *Transactions*,	3,458.00	
Binding miscellaneous volumes,	256.70	
Engraving and electrotyping,	780.31	
Secretary's department, including clerks, stenographers, and expenses of editing and proof-reading, and special assistance in connection with meetings,	10,633.67	
Treasurer's department, including collection of dues, shipping, etc.,	6,473.67	
Librarian and assistants,	1,484 54	
Postage,	3,844 12	
Stationery,	545 88	
Express and freight charges,	1,719 48	
Telephone,	245.45	
Telegrams, cables, carfares, etc.,	74.34	
Office supplies and repairs,	121.57	
Refunding miscellaneous payments,	38.87	
Insurance premiums (Fire and Surety),	204.65	
Collection charges,	34.14	
Extra clerical assistance,	27.42	
Special stenographers and expenses of meetings,	1,466.96	
Auditing,	125.00	
Office cleaning and sundry expenses,	39.76	
		$46,350.39
Interest at 4 per cent., for 1910, on unpaid balance of land mortgage on 25 to 33 West 39th St. ($88,000, January 1, 1910, reduced to $85,000 January 1, 1911),	3,520.00	
Quota of current expenses of building 25 to 33 West 39th St.,	4,500.00	
		8,020.00
Special editing, part payments on printing and binding special edition, new volume of Genesis of Ore-Deposits,		110.55
Library additions of books, periodicals, etc., binding of exchanges, and stationery (expenditure from appropriation of $2,500.00),		1,163.63
Furniture and Fixtures,		260 57
Balance,		3,938.17
		$59,843.31

NEW YORK, N. Y., January 21, 1911.

We have examined the above statement, compared it with the books and vouchers and find same correct.

(Signed) BARROW, WADE, GUTHRIE & CO.,
Certified Public Accountants.

* $17,045 of this amount has been applied to subscriptions to the *Bulletin* in accordance with post-office regulations.

REPORT OF THE COUNCIL FOR THE YEAR 1910.

The following acts of the Council are here published for the information of members:

At the meeting of the Council, Dec. 9, 1910, Prof. Tsunashiro Wada, of Tokyo, Japan, was unanimously recommended to the Board of Directors for election as an Honorary Member of the Institute in recognition of his eminent services to the sciences and industries represented by the Institute.

Mr. H. D. Hibbard was appointed to represent the Institute on Committee No. 24 of the International Association for Testing Materials, on the nomenclature of iron and steel.

On Feb. 21, 1911, Messrs. W. L. Saunders and George C. Stone were appointed delegates to the Eighth Session of the International Congress of Applied Chemistry, New York, September, 1912.

Committee on Membership: (to serve during ensuing year) Benjamin B. Lawrence, Karl Eilers, Charles F. Rand, Edward L. Young, and R. W. Raymond.

Institute Meetings.

There were two meetings of the Institute held during the year 1910 for the reading and discussion of papers—the Ninety-eighth Meeting, in Pittsburg, Mar. 1 to 5, and the Ninety-ninth Meeting, and excursions, in the Canal Zone, Oct. 21 to Nov. 15.

A detailed record of the proceedings of these meetings, including a description of the entertainments and excursions connected therewith, has been published and duly distributed to the members: the Pittsburg meeting in *Bulletin* No. 40, April, 1910, pp. 311 to 334, and the Canal Zone meeting in *Bulletin* No. 48, December, 1910, pp. 1017 to 1054. At the Pittsburg meeting there were presented 48 papers and 12 discussions, oral and written; in these discussions 15 separate contributors participated. At the Canal Zone meeting there were presented 49 papers and 9 discussions, oral or written; in these discussions 65 contributors participated. At the Pittsburg meeting the names of 155 members and guests were registered at the Institute headquarters; this number, however, does not represent all who were present at the sessions and the excursions. In connection with the Canal Zone meeting, the num-

ber of members and guests comprising the Institute party on the steamer *Prinz August Wilhelm* was 122, but the total number participating in the excursions, in Havana, Kingston, and the Canal Zone, or attending, in whole or in part, the sessions at Ancon, exceeded 300.

PUBLICATIONS.

Transactions.—Volume XL. of the *Transactions*, an octavo of 1,002 pages, comprising 50 papers and 17 discussions presented during the year 1909, was issued and distributed to members in June, a little earlier in the year than the corresponding appearance of Volume XXXIX. Most of the material for Volume XLI., forming in all about 1,000 pages, is in the hands of the printer, and it is expected that the bound volume will be off the press and ready for distribution in June, 1911.

Bulletin.—Twelve numbers of the *Bulletin* (Nos. 37 to 48), containing the technical papers and discussions of the Institute (in "subject to revision" form) and announcements of general interest to the members of the Institute, such as Library accessions and requirements during the year 1910; notices of meetings of the Institute and of other societies; lists of proposed members and associates; changes of address; deaths of members; obituary notices; Index of Titles and Authors, etc., have been published and distributed promptly throughout the year 1910. The number of pages occupied by technical papers and discussions amounts to 1,066, to which are to be added 340 pages of announcements, and 272 pages of advertising matter, making a total of 1,678 pages of printed matter.

The editorial and business management of the *Bulletin*, Volume XL. and the forthcoming Volume XLI. of the *Transactions* continues in charge of Dr. Joseph Struthers, Assistant Secretary and Editor of the Institute.

MEMBERSHIP.

Changes in membership have taken place during the year as follows: 1 honorary member, 170 members, and 5 associates have been elected; 3 members have been reinstated; 4 associates have become members; the deaths of 47 members and 1 associate have been reported; 85 members and 5 associates have resigned; and 112 members and 3 associates have been dropped from the roll by reason of non-payment of dues, loss

of correct address, etc.* These changes are shown in the accompanying table.

The total membership on Jan. 1, 1911, was 4,210, as compared with 4,284 on Jan. 1, 1910.

Membership of the American Institute of Mining Engineers, Jan. 1, 1911.

	Honorary Members.	Honorary Associates	Members	Associates	Totals.
Membership Dec 31, 1909	14	1	4,111	158	4,284
Gains: By Election	1		170	5	176
Change of Status			4		4
Reinstatement			3		3
Losses: By Resignation			85	5	90
Change of Status				4	4
Dropping			112	3	115
Death			47	1	48
Total gains	1		177	5	183
Total losses			244	13	257
Membership Dec. 31, 1910	15	1	4,044	150	4,210

The list of deaths reported during the year 1910 comprises the following names, the figures in parentheses indicating the year in which the persons named were elected to membership:

Members and Associates.—Masayoshi Abe (1905), W. Edward Adams (1903), James Archbald (1887), John H. Bartlett (1890), William F. Biddle (1881), William P. Blake (1871), Wager Bradford (1902), Arthur Brock (1887), Fayette Brown (1895), Henry Burrell (1900), José Calero (1901), Frank J. Campbell (1899), Frank R. Carpenter (1887), Octave Chanute (1879), E. W. Codington (1890), R. Prewitt Coleman (1903), Francis V. Drake (1907), Charles H. Ferry (1891), James W. Fuller (1894), Paul A. Fusz (1879), Edward C. Hegeler (1881), Gus C. Henning (1886), A. D. Hodges, Jr. (1884), John W. Hoffman (1876), Ottokar Hofmann (1884), Thomas A. Irvin (1906), Guy R. Johnson (1889), Alfred Kimber (1907), Herbert H. Light (1905), Edmund D. North (1902), Josiah Owen (1900), Charles B. Parsons (1874), Ernest Y. Pomeroy (1906), Pietro Redaelli (1905), Ferd H. Regel (1900), William H. Schlemm (1888), John C. Sevier (1906), H. A. Shipman (1899), Albert Spies (1881), Herbert S. Stark (1897), John Sutcliffe (1887), James P. Wallace (1897), Thomas F. Walsh (1900), S. Bowman Wheeler (1892), Wilfred F. Wheeler (1907), Frederick de L. Williams (1899), A. B. Wood (1882), Alfred F. Wuensch (1900).

* Many of these, no doubt, will be reinstated, as has been the case in former years.

MEMBERSHIP.

The following list comprises the names of those persons elected as members, who duly accepted election during the year 1911. The marks used to designate the different classes of membership are: Life Member, **; Member,*; Associate Member, †. Heavy-faced type signifies Honorary Membership.

*Apgar, Frederick W., Jamaica, N. Y.
*Archbald, Hugh, Scranton, Pa.
*Bailey, A. C., Cobalt, Ont., Canada.
*Barker, George, Rosebery, Tasmania.
*Beeken, Lewis L., Pittsburg, Pa.
*Binford, Charles M, Stanaford, W. Va.
*Borie, Adolph E., New York, N Y.
*Bowen, David, Leeds, England.
*Bowler, Robert P, New York, N. Y.
†Bridgman, John C., Wilkes-Barre, Pa.
*Brindle, A. C., Victoria, B. C., Canada.
*Brodrick, Carlton T., Kyshtim, Perm Govt, Russia.
*Brown, A. L., Wallaroo, So. Australia.
*Brown, Charles H., Magdalena, N. M.
*Browne, Spencer C., Oakland, Cal.
**Brunton, F. K., Anaconda, Mont.
*Bryden, Alexander, Dunmore, Pa.
*Buchanan, Jerome R, Bodie, Cal.
*Burch, Henry K., Globe, Ariz.
*Bush, Morris W., Woodward, Ala.
*Cahoone, William M., Benton, Cal.
*Carlyle, Ernest J., Kyshtim, Perm Govt., Russia.
*Cavazos, E., Saltillo, Coah., Mexico.
*Chadbourne, Humphrey W., West Palm Beach, Fla.
*Chance, Edwin M., Pottsville, Pa.
*Chartier, George M., Los Angeles, Cal.
*Chase, Fred M., Wilkes-Barre, Pa.
*Clark, John E., Riverside, Cal.
*Clarke, Alexander C, Midgham House, near Reading, England.
*Corbin, James R, Philadelphia, Pa.
*Cox, Guy H., Rolla, Mo.
*Crabtree, Fred, Pittsburg, Pa.
*Cuellar, Salvador, Ojenaga, Chih., Mex.
*Daniels, Joseph, South Bethlehem, Pa.
*Davis, Henry G., Kingston, Pa.
*Davis, John A., Washington, D. C.
†Deming, Henry C., New York, N. Y.
*Devereux, W. B, Jr., New York, N. Y.
*Dixon, Abner F., Bombay, India.
*Dobbs, Gerald G., Bessemer, Ala.
*Dodge, David C., Denver, Colo.
*Dodge, William F., Wilkes-Barre, Pa.
*Dorrance, Charles, Jr., Lansford, Pa.
*Duck, George F., Denver, Colo.
*Dull, A. J., Harrisburg, Pa.
*Duncan, G. S, London, E. C., England.
*Dunstan, S. P., Oyon, Peru, So. Am.
*Durkee, F. W., Tufts College, Mass.
*Dutton, Charles E, Goldfield, Nev.
*Earling, Roy B., Ray, Ariz.
*Edelsteen, Karl J., Exeter, Cal.
*Ederheimer, Leopold, New York, N.Y.
*Emmel, Rudolph, Boston, Mass.
*Engel, George W., Scranton, Pa.
*Enzian, Charles, Wilkes-Barre, Pa.
*Eu, Siang Hye, Nanking, China.
*Fenner, Charles H, Los Angeles, Cal.
*Fisher, Howell T., Germantown, Pa.
*Foster, D. F., San Julian, Chih., Mex.
*Fraser, Lee, Ormo, Bolivia, So. Am.
*Fukitome, Kinosuke, Formosa Govt., Taipeh, Japan.
*Gard, I. R., Victoria, B. C, Canada.
*Gayford, Ernest, Salt Lake City, Utah.
*Gennet, Charles W., Jr., Chicago, Ill.
*Gibbons, C. A., Zimapan, Hid., Mexico.
*Goldsworthy, J., Vancouver, B. C., Can.
*Goode, Ewart N., Port Kembla, N.S.W., Aust.
**Gordon, A.R., San Juancito, Honduras.
*Griffith, William, Scranton, Pa.
*Grover, M. B., Haileybury, Ont., Can.
*Hamilton, E. H., West Norfolk, Va.
*Hansen, Fred, Garfield, Utah.
*Harada, Shinji, Tokyo, Japan.
*Harris, A. L., Agujita, Coah., Mexico.
*Hart, V. A., Cananea, Son., Mexico.
*Hasegawa, Kanji, Kagoshima, Japan.

*Heimer, P. H., Porcupine, Ont , Can.
*Henderson, Charles W., Denver, Colo.
*Herrmann, Charles E , New York, N.Y.
*Hoffmann, A. O , Meamorskaja, Russia.
*Hopper, Walter E., Madison, Wis.
*Hotchkin, M W., Haileybury, Ont., Can.
*Houck, Charles B., Hazleton, Pa.
*Howell, Franklin D., Los Angeles, Cal.
*Hower, Charles L , Spokane, Wash.
†Huang, Saosan Ken, So. Bethlehem, Pa.
*Huber, Charles F., Wilkes-Barre, Pa.
*Hunter, C., Pilgrims Rest, Transvaal, So. Africa.
*Hutcheson, W. C., Belle Ellen, Ala.
*Inouye, Koji, Shimotsuke, Japan.
*Ives, Glen P., Illapel, Chile, So. Am.
*James, William E., Carbon, W. Va.
*Jeffreys, G., Tampico, Tamps., Mexico.
*Jewett, Freeland, Boston, Mass.
*Johnson, E. H., East Rand, Transvaal, So. Africa.
*Kane, John I., El Paso, Texas.
*Kano, Shinichi, Osaka, Japan.
†Kennedy, Arthur T., Kinney, Minn.
*Kennedy, Joseph E., New York, N. Y.
*Kenney, Robert M., Golden, Colo.
*Kepner, Ross B , Sierra, Coah., Mexico.
*Kiddie, John, Morenci, Ariz.
*Ko, Sokichi, Fukuota, Japan.
*Kohlbraker, F. H , Nanticoke, Pa.
*Kramm, Hugo E., Ithaca, N. Y.
*Kruemmer, A. W , New York, N. Y.
*La Croix, Morris F., Ishpeming, Mich.
*Lanagan, William H., Nikolaievsk-on-Amur, E. Siberia.
*Law, A. F., Scranton, Pa.
*Leckie, J. E., Cobalt, Ont., Canada.
*Leisenring, A. C., Upper Lehigh, Pa.
*Le Noir, F. H., Mount Bullion, Cal.
*Linton, R. A., Tuquerres, Colombia, So. America.
*Lippincott, J. B., Los Angeles, Cal
*Locke, Augustus, Hampton, N. H.
*Logan, Spencer R., Telluride, Colo.
*London, Clarence J., Philadelphia, Pa.
*Loomis, Willis H., Jeddo, Pa.
*McCosh, A. K., Coalbridge, Scotland.
*McMahon, F. J., Wilkes-Barre, Pa.
*McRandle, W. E., Bessemer, Mich.
*Macauley, Rupert M., Copper Cliff, Ont., Canada.
*Manahan, Robert F., Cambridge, Mass.
*Mansfield, Melvin, Salt Lake City, Utah.
*Marquard, William B., Easton, Pa.
*Master, N. M., Ipoh, Perak, F. M. S.
*Matsukata, Otohiko, Echigo, Japan.
*Mavor, Sam, Glasgow, Scotland.
*Maynard, Thomas P., Atlanta, Ga.
*Menefee, Arthur B , Wharton, N. J.
*Merrill, Monroe E., Hollywood, Cal.
*Meyerovitch, Joseph A , St. Petersburg, Russia.
*Miller, B. Le R., South Bethlehem, Pa.
*Mills, Kenneth, Jacala, Hid., Mexico.
*Montgomery, Ernest A., Los Angeles, Cal.
*Mostowitsch, Wladimir, Riga, Russia.
*Moxham, Arthur J., Wilmington, Del.
*Murota, Yashibumi, Tokyo, Japan.
*Naito, Hisahiro, Echigo, Japan.
*Newell, G. S., Matehuala, S.L.P., Mex.
†Nicholson, S. T., Wilkes-Barre, Pa.
*Nighman, C E., Silver Centre, Ont., Can.
*Nishimura, K., Fimatsu, Hida, Japan.
*Opie, N., Wallaroo Mines, So. Australia.
*Orbison, Thomas W., Appleton, Wis.
*Palmer, Irving A., Springfield, Ill.
*Parry, C. F., Germiston, Transvaal, So. Africa.
*Peale, Rembrandt, New York, N. Y.
*Peck, Walter R , Big Stone Gap, Va.
*Penhallegon, W. J., Birmingham, Ala.
*Pettebone, Edgar R., Scranton, Pa.
*Playter, Joseph H., Golconda, Nev.
†Prince, Ernest, Chicago, Ill.
*Quin, Robert A., Wilkes-Barre, Pa.
*Radcliffe, Alfred, Copiapo, Chile, S. A.
*Randall, David V., Minersville, Pa.
†Rehfuss, Louis A., Telluride, Colo.
*Richard, G. M., Latouche, Alaska.
*Roeber, E. F., New York, N. Y
*Rogers, William B., New York, N. Y.
†Russell, Charles M., Massillon, Ohio.
*Sacket, Charles T., Livingston, Mont.
†Sahlin, Robert C., Bethlehem, Pa.
*Sanders, B. H., Cartagena, Colombia, So. America.
*Scaife, Hazel L., Clinton, S. C.
*Scheble, Max C., Lampacitos, Coah., Mexico.
*Schwennesen, Alvin T., Clayton, Cal.
*Shaw, Alexander J. M., Chiao Tso, Honan, No. China.
*Shutts, A. B., Manillas, Zac., Mexico.
*Simpson, Kenneth M., Reno, Nev.

*Sinn, Francis P., Palmerton, Pa.
*Sirdevan, W. H., San Francisco, Cal.
*Slee, W E., Wallaroo, So. Australia.
*Smith, Henry P , Guanajuato, Mexico.
*Smith, Sumner S., Juneau, Alaska.
*Spicer, Philip O., Kelowna, B C, Can.
*Squires, Howard W., Los Angeles, Cal.
*Sterling, Paul, Wilkes-Barre, Pa.
*Stevens, Arthur W., Atlanta, Idaho.
*Stevenson, George E., Scranton, Pa.
†Stillman, James S., Catasauqua, Pa.
*Storrs, Arthur H., Scranton, Pa.
*Tainter, F. S., Hoboken, N. J.
*Takenouchi, Korehiko, Prov. Hitachi, Japan.
*Thayer, Reginald H, Yonkers, N. Y.
*Thomas, Charles S., Jr., Denver, Colo.
*Thomas, Edmund, Dawson, N. M.
*Tryon, Charles T., Boston, Mass.
*Van Horn, Frank R., Cleveland, Ohio.
*Verrill, C. S , Vancouver, B. C., Can.
*Waite, Henry M., Dante, Va.
*Warren, Oscar Bird, Hibbing, Minn.
†Weaver, Henry M., Mansfield, Ohio.
*Wentworth, Henry A., Boston, Mass.
*Whitehead, Harry F., Inman, Va.
*Whittier, Charles C , Chicago, Ill.
*Wood, Richard G., Conshohocken, Pa.
*Wolf, Artin Y., San Diego, Cal.
†Woollcombe, R L, Dublin, Ireland.
†Yates, James, Sublet, Wyo.
*Zapffe, Carl, Brainerd, Minn.
*Zoffman, G. F., Guanajuato, Mexico.

Deaths.

The following list comprises the names of members whose deaths have been reported to the Secretary of the Institute during the year 1911:

Date of Election.	Name.	Date of Decease.
1906.	*Affleck, W.,	Sept. 2, 1911.
1905.	*Alabaster, R. C.,	Feb. 12, 1911.
1899.	*Alberger, L. R,	Jan, 31, 1911.
1903.	*Bamberger, S. M.,	May 10, 1911.
1901.	*Briggs, R. E.,	May 5, 1911.
1908.	*Brill, Paul K.,	Mar. 3, 1911.
1875.	*Brown, A. E.,	Apr. 26, 1911.
1879.	*Bulkley, H. W.,	Nov. 7, 1911.
1876.	*Chouteau, P.,	Nov. 21, 1910.
1882.	*Collingwood, F.,	Aug. 18, 1911.
1903.	*Cosby, Robert P.	Apr. —, 1910.
1906.	*Culbert, M. T.,	Mar. 14, 1911.
1897.	*Diggles, J. A.,	May 14, 1910.
1881.	*Dods, John C.,	Sept. 1, 1911.
1877.	**Emmons, S. F.,	Mar. 28, 1911.
1903.	*Emrich, H. H.,	Oct. 18, 1911.
1902.	**Forrester, R.,	Dec. 20, 1910.
1898.	*Grave, Percy,	Jan. 22, 1911.
1896.	*Grillo, Julius,	Mar. —, 1911.
1887.	*Grubb, C. B.,	Nov. 12, 1911.
1890.	*Hesse, C. E.,	May 30, 1910.
1903.	*Holmes, E. M.,	Feb. 11, 1911.
1886.	†Howe, E.,	Jan. 25, 1911.
1895.	†Hughes, C. J.,	Jan. 11, 1911.
1891.	**Hunt, C. W.,	Mar. 27, 1911.
1872.	**Janin, Henry,	Jan. 6, 1911.
1880.	*Johnson, J. E.,	Apr. 30, 1911.
1881.	**Jones, W.,	July 30, 1911.
1893.	*Kurtz, H. M.,	Mar. 30, 1911.
1893.	*Lawrence, H. L.,	May 8, 1911.
1890.	*Lee, J. H.,	Jan. 25, 1911.
1875.	*Lord, N. W.,	May 22, 1911.
1903.	*McCan, E. K.,	Oct. 29, 1910.
1898.	*McClurg, J. A.,	May 4, 1910.
1881.	**Martin, E. P.,	Sept. 25, 1910.
1897.	*Matcham, C. A.,	Sept. 22, 1911.
1891.	**Metcalf, A. T.,	Nov. 29, 1910.
1874.	*Morgan, C. H.,	Jan. 10, 1911.
1896.	*Murphy, T. D.,	Apr. 3, 1911.
1909.	*Norbom, J. O.,	Jan. 13, 1911.
1890.	†Norrie, A. L.,	Dec. 22, 1910.
1882.	*Potts, F. L.,	Mar. 11, 1910.
1879.	*Richards, E. H,	Mar. 30, 1911.
1909.	*Shelby, C. F.,	Jan. 25, 1911.
1885.	*Sticht, Ernest,	Mar. 12, 1911.
1899.	**Sutherland, W. J.,	Ap. 22, 1911.
1904.	**Swan, A. A.,	Feb. 12, 1911.
1876.	*Thompson, H. S.,	Mar. 9, 1911.
1876.	*Valentine, M. D.,	July 4, 1911.
1909.	*Weiss, R. A.,	July 11, 1911.
1888.	*Wood, Howard,	July 1, 1911.

* Member. ** Life Member. † Associate.

Proceedings of the One Hundredth Meeting, Wilkes-Barre, June, 1911.

LOCAL COMMITTEES.

EXECUTIVE.—W. A. Lathrop, *Chairman;* R V. Norris, *Secretary,* S. D. Warriner, *Treasurer;* Irving A. Stearns, W. J. Richards, H. S. Drinker, C. D. Simpson.

GENERAL RECEPTION.—Irving A. Stearns, *Chairman.*

Archibald, James, Jr., Pottsville.
Ashley, H. H., Wilkes-Barre.
Ayres, W. S., Hazleton.
Beard, J. T., Scranton.
Bridgman, J. C., Wilkes-Barre.
Bunting, Douglas, Wilkes-Barre.
Chase, F. M., Wilkes-Barre.
Conner, Eli T., Scranton.
Coxe, E. B., Jr., Drifton.
Davies, W. H., Hazleton.
Davis, H. G., Dorranceton.
Dodge, W. F., Wilkes-Barre.
Drinker, H. S., South Bethlehem.
Emmerich, L O., Hazleton.
Enzian, Charles, Wilkes-Barre.
Foster, R. J., Scranton.
Fritz, John, Bethlehem.
Hill, F. A., Pottsville.
Houck, C. B., Hazleton.
Huber, C. F., Wilkes-Barre.
Humphrey, John M., Centralia.
Jessup, A. B., Wilkes-Barre.
Jones, J. E, New Boston.
Jones, T. D., Hazleton.
Lathrop, W. A., Wilkes-Barre.
Lawall, E. H., Wilkes-Barre.
Lentz, W. O., Mauch Chunk
Lewis, Albert, Bear Creek.
Loomis, W. H., Jeddo.
Markle, Alvan, Hazleton.
Markle, John, Jeddo.
Neale, J. B., Minersville.
Norris, R. V., Wilkes-Barre.
Oliver, Paul A., Oliver's Mills.
Owens, W. D., Pittston.
Pardee, I. P., Hazleton.
Quin, R. A., Wilkes-Barre.
Richards, W. J., Pottsville.
Righter, T. M., Mount Carmel.
Simpson, C D., Scranton.
Snyder, Baird, Jr., Lansford.
Storrs, A. H., Scranton.
Straw, C. A., Lansford.
Sturges, C. B., Scranton.
Thomas, Thomas, Wilkes-Barre.
Warriner, S. D., Wilkes-Barre.
Welles, T. L., Wilkes-Barre.
Whildin, W. G., Lansford.
Wolf, T. G., Scranton.
Zerbey, F. E, Wilkes-Barre.

The first session, held Tuesday evening, June 6, in the ballroom of the Glen Summit Springs Hotel, was called to order by W. A. Lathrop, Chairman of the Local Executive Committee. Mr. Lathrop, on behalf of the many friends of the Institute in the anthracite region, extended a cordial welcome to the members and guests present. Charles Kirchhoff, President of the Institute, who presided at the meeting, responded for the Institute.

A letter of hearty congratulations on the one hundredth meeting of the Institute, was received from the Verein deutscher Eisenhüttenleute. This testimonial, in German text, beautifully illuminated on parchment and bound in leather, is translated as follows:

HONORED MR. PRESIDENT:

The One Hundredth Meeting of your Institute is an occasion welcome to us, to present to your Society and to its members heartiest congratulations for this festive day, and to express our high appreciation of the admirable achievements which the American Institute of Mining Engineers may look back upon with justifiable pride, after an existence of 40 years. Through your many activities, through the practical and scientific work of your members, you have to a marked degree contributed to the successful development of the enormous mineral wealth of your country, and to its metallurgical utilization.

You have at the same time proved that technology and science are international, and by your work have contributed to the unexampled rise of mining and metallurgy in all countries during the last decades. We remember gratefully the friendly relations which have existed for many years between our society and yours, and which have been expressed through repeated successful joint meetings of the societies, and through cordial personal relations of the members of the two societies. In expressing the hope that the friendly relations between the two societies may continue in the future, as in the past, we remain, with repeated hearty congratulations on this festive day, and with joyous "Glückauf" for the future of your society,

VEREIN DEUTSCHER EISENHÜTTENLEUTE.

Presiding Officer:
SPRINGORUM,
Konigl. Kommerzienrat.

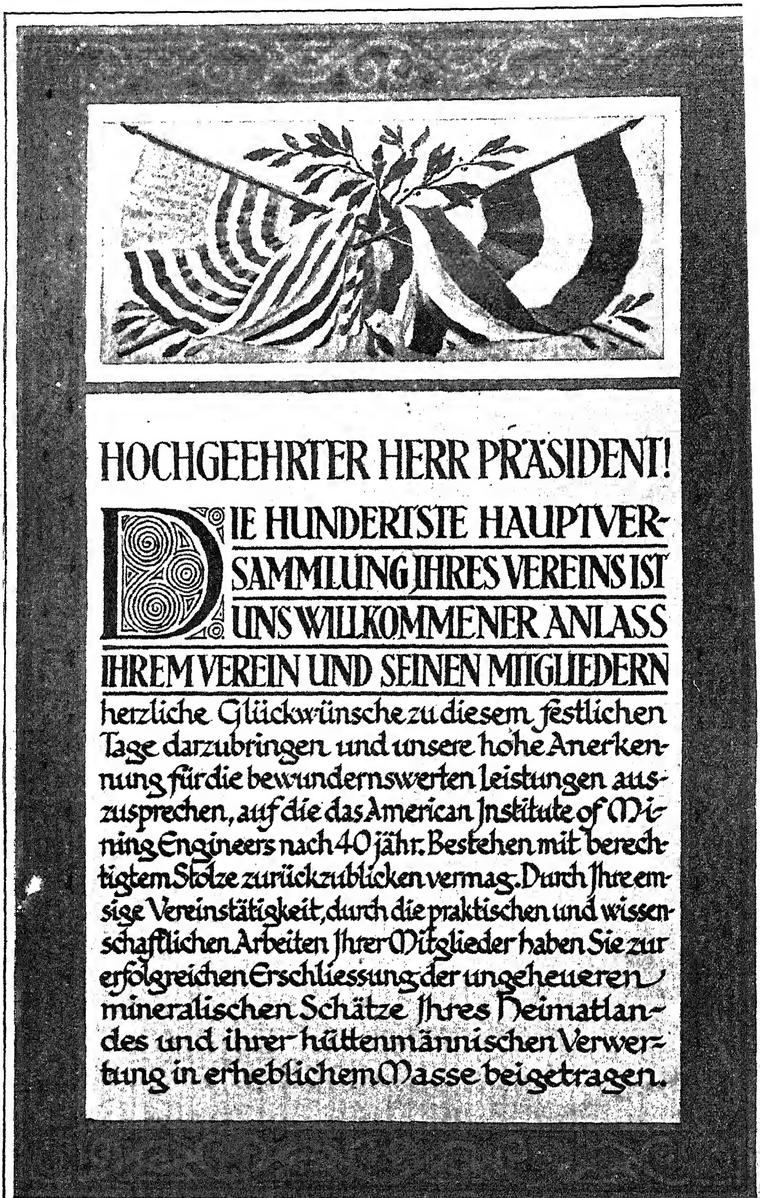
Secretary:
E. SCHROEDTER.

Düsseldorf, May, 1911.

American Institute of Mining Engineers, by Hand of Presiding Officer CHARLES KIRCHHOFF.

Dr. Henry S. Drinker, President of Lehigh University, presented the following message from Mr. John Fritz (Uncle John Fritz), of Bethlehem, who had fully intended to be at Glen Summit, but was at the last moment obliged to give up the trip on the advice of his doctor. Mr. Fritz's message was as follows, as Dr. Drinker took it down from his lips before leaving Bethlehem:

"I meant to be with you and am sorry, very sorry I cannot be with you in person, but I am with you in spirit now and forever."



HOCHGEEHRTER HERR PRÄSIDENT!

DIE HUNDERTSTE HAUPTVERSAMMLUNG IHRES VEREINS IST UNS WILLKOMMENER ANLASS IHREM VEREIN UND SEINEN MITGLIEDERN herzliche Glückwünsche zu diesem festlichen Tage darzubringen und unsere hohe Anerkennung für die bewundernswerten Leistungen auszusprechen, auf die das American Institute of Mining Engineers nach 40 jähr. Bestehen mit berechtigtem Stolze zurückzublicken vermag. Durch Ihre emsige Vereinstätigkeit, durch die praktischen und wissenschaftlichen Arbeiten Ihrer Mitglieder haben Sie zur erfolgreichen Erschliessung der ungeheueren mineralischen Schätze Ihres Heimatlandes und ihrer hüttenmännischen Verwertung in erheblichem Masse beigetragen.

ILLUMINATED LETTER OF CONGRATULATION FR

SIE HABEN DABEI GLEICHZEITIG dokumentiert, dass Technik und Wissenschaft international sind, und durch Ihre Tätigkeit an dem beispiellosen Aufschwung des Berg- und Hüttenwesens aller Länder in den letzten Jahrzehnten mitgewirkt. Wir erinnern uns heute dankbar der freundschaftlichen Beziehungen, in welchen unser Verein zu dem Ihrigen schon seit langen Jahren steht und die durch mehrfache erfolgreiche Zusammenkünfte der Vereine wie lebhafte Einzelbeziehungen der Mitglieder der beiden Vereine zum Ausdruck gekommen sind. Indem wir der Hoffnung Ausdruck verleihen, dass das freundschaftliche Verhältnis zwischen den beiden Vereinen, das gerade durch Ihre Person verkörpert wird, auch in Zukunft wie bisher obwalten werde, verbleiben wir mit nochmaligen herzlichen Glückwünschen zu dem Festtage und mit frohem Glückauf für die Zukunft Ihres Vereins Ihr ergebenster

VEREIN DEUTSCHER EISENHÜTTENLEUTE:

Der Vorsitzende: Springorum. Königl. Kommerzienrat:

Der Geschäftsführer: E. Schrödter

Düsseldorf, im Mai 1911.

AMERICAN INSTITUTE OF MINING ENGINEERS

z. H. des Vorsitzenden HERRN CHARLES KIRCHHOFF

THE VEREIN DEUTSCHER EISENHÜTTENLEUTE.

The following papers were presented in oral abstract by the authors:

*The Storage of Anthracite Coal, by R. V. Norris, Wilkes-Barre, Pa. (Discussion by Charles P. Perin, New York, N. Y.[1])

*The Preparation of Anthracite, by Paul Sterling, Wilkes-Barre, Pa.

The Summit Hill Mine-Fire, by W. A. Lathrop, Philadelphia, Pa.[2] (Illustrated by lantern-slides.)

Reminiscences of the Beginning of the Institute, by R. W. Raymond, New York, N. Y.[2]

The second session, held in the rooms of the Wyoming Historical and Geological Society, Wilkes-Barre, Wednesday afternoon, June 7, was called to order by President Kirchhoff.

Major Irving A. Stearns, President of the Society, cordially welcomed the members and guests, and President Kirchhoff, on behalf of the Institute, responded.

The President announced that, upon the proposal of many members and the unanimous recommendation of the Council, the following members had been unanimously elected by the Board of Directors as Honorary Members of the American Institute of Mining Engineers:

Prof. Robert H. Richards, Boston, Mass., and Dr. Rossiter W. Raymond, New York, N. Y.

The following papers were presented in oral abstract by the authors:

*The Anthracite Board of Conciliation, by S. D. Warriner, Wilkes-Barre, Pa. (Discussion by E. W. Parker, Washington, D. C., and D. B. Rushmore, Schenectady, N. Y., and reply by Mr. Warriner.[1])

*The United States Iron Industry from 1871 to 1910, by John Birkinbine, Philadelphia, Pa.

The third and concluding session was held on Thursday evening, June 8, in the ball-room of the Glen Summit Springs Hotel; President Kirchhoff presided.

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* Distributed in printed form.

[1] Discussion not furnished for publication.

[2] Not furnished for publication.

The following papers, illustrated by lantern-slides, were presented in oral abstract by the authors:

Mine-Caves Under the City of Scranton, by Eli T. Conner, Scranton, Pa.

Materials Available for Refilling Coal-Workings in the Northern Anthracite Coal-Field, by N. H. Darton, Washington, D. C.[3]

Mine-Rescue Service of the State of Illinois, by H. H. Stoek, Urbana, Ill.

Electric Motors versus Compressed-Air Engines for Driving Deep-Mine Hoists, by K. A. Pauly, Schenectady, N. Y.

*The Sintering of Fine Iron-Bearing Materials, by James Gayley, New York, N. Y. (In the absence of Mr. Gayley, this paper was presented by Arthur S. Dwight, New York, N. Y. Discussion by James E. Little, Steelton, Pa., and Benjamin W. Vallat, Ironwood, Mich.[4])

In addition to the papers already noted, the following were read by title for future publication:

*Geology of the Cobalt District, Ontario, Canada, by Reginald E. Hore, Houghton, Mich.

*Origin of Certain Bonanza Silver-Ores of the Arid Region, by Charles R. Keyes, Des Moines, Iowa.

*Assay of Silver-Bearing Gouge-Ores, by Charles R. Keyes, Des Moines, Iowa, and D. F. Riddell, Parral, Mexico.

*A Drafting-Table for Tracing Through Opaque Paper, by A. T. Schwennesen, Stanford University, Cal.

*Lead-Smelting in the Ore-Hearth, by J. J. Brown, Jr., Wilburton, Okla.

*The Caddo Oil- and Gas-Field, Louisiana, by Walter E. Hopper, Madison, Wis.

*Origin of the Iron-Ores of Central and Northeastern Cuba, by C. K. Leith and W. J. Mead, Madison, Wis.

*Occurrence, Origin, and Character of the Surficial Iron-Ores of Camaguey and Oriente Provinces, Cuba, by Arthur C. Spencer, Washington, D. C.

*The Mayari and Moa Iron-Ore Deposits in Cuba, by C. Willard Hayes, Washington, D. C.

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* Distributed in printed form.

[3] Not furnished for publication.

[4] Discussion not furnished for publication.

*Exploration of Cuban Iron-Ore Deposits, by Dwight E. Woodbridge, Duluth, Minn.

*The Iron-Ore Deposits of the Moa District, Oriente Province, Island of Cuba, by Jennings S. Cox, Jr., Santiago de Cuba, Cuba.

*Characteristics and Origin of the Brown Iron-Ores of Camaguey and Moa, Cuba, by Willard L. Cumings and Benjamin L. Miller, Bethlehem, Pa.

*The Fuel-Efficiency of the Iron Blast-Furnace, by John Jermain Porter, Cincinnati, Ohio.

The Continuous System of Cyaniding in Pachuca Tanks, by Huntington Adams, Natividad, Oaxaca, Mexico.

*Mining-Costs at Park City, Utah, by Fred T. Williams, Park City, Utah.

*Diagonal-Plane Concentrating-Table, by S. Arthur Krom, Plainfield, N. J.

History and Geology of Ancient Gold-Fields in Turkey, by Leon Dominian, New York, N. Y.

*Tunnel-Driving in the Alps, by W. L. Saunders, New York, N. Y.

Anthracite-Culm Briquettes, by Charles Dorrance, Jr., Lansford, Pa.

*Canadian Mining-Law, by J. M. Clark, Toronto, Canada, and Discussion by Dr. R. W. Raymond, New York, N. Y.

Loss in "Breaking Down" Anthracite, by W. F. Dodge, Wilkes-Barre, Pa.[5]

Apparatus for Metallography, by Carle R. Hayward, Boston, Mass.

The Universal Metalloscope, by Albert Sauveur, Cambridge, Mass.

The Preparation of Brown Iron-Ores, by H. S. Geismer, Chattanooga, Tenn.

Treatment of Nicaraguan Gold-Ores, by Henry B. Kaeding, Nicaragua, C. A.

Structure of the Northern Anthracite Coal-Field, Especially in Relation to the Occurrence of Gas in the Coal, by N. H. Darton, Washington, D. C.[5]

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* Distributed in printed form.

[5] Not furnished for publication.

*Chamber-Pillars in Deep Anthracite-Mines, by Douglas Bunting, Wilkes-Barre, Pa.

Use of Electricity in Anthracite-Mining, by David B. Rushmore, Schenectady, N. Y.[6]

Notes on Huntington Mills in Nicaragua, by Clarence C. Semple, New York, N. Y.

Mine Rescue-Work in Illinois, by J. A. Holmes, Washington, D. C.[6]

The Mayari Iron-Mines, Oriente Province, Island of Cuba, as Developed by the Spanish-American Iron Co., by James E. Little, Steelton, Pa.

Discussion of the paper of G. W. Riter, Mine-Survey Notes, by E. R. Rice, Wickensburg, Ariz.

*Discussion of the paper of W. H. Emmons, The Agency of Manganese in the Superficial Alteration and Secondary Enrichment of Gold-Deposits in the United States, by Charles R. Keyes, Des Moines, Iowa.

*Discussion of the paper of William Wraith, Sampling Anode-Copper, with Special Reference to Silver-Content, by Edward Keller, Perth Amboy, N. J.

Discussion of the paper of R. E. Hore, Geology of the Cobalt District, Ontario, Canada, by C. W. Knight, Toronto, Ontario, Canada.

Discussion of the paper of Eli T. Conner, Mine-Caves Under the City of Scranton, by R. J. Foster, Scranton, Pa.

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* Distributed in printed form.

[6] Not furnished for publication.

Excursions and Entertainments.

An account of the excursions and entertainments in connection with the Wilkes-Barre meeting was published in *Bulletin* No. 55, July, 1911, pp. 584 to 592.

Members and Guests in Attendance at the Sessions and Excursions.

Adams, G. T., Washington, D. C.
Archibald, Hugh, Scranton, Pa.
Archibald, J., Jr., Scranton, Pa.
Ayres, Mrs. E. L. C., Bound Brook, N. J.
Ayres, W. S , Hazleton, Pa.
Baelz, W., New York, N. Y.
Beard, H. I., Scranton, Pa.
Beard, J. T., Scranton, Pa.
Beard, J. T., Jr., Scranton, Pa.
Benjamin, E. H., Oakland, Cal.
Bird, R. M , South Bethlehem, Pa.
Birkinbine, J., Philadelphia, Pa.
Birkinbine, J. L. W., Philadelphia, Pa.
Bowler, R. P., New York, N. Y.
Boyd, H., Hokendauqua, Pa.
Bridgman, J. C., Wilkes-Barre, Pa.
Bryden, A., Dunmore, Pa.
Bryden, C. L., Scranton, Pa.
Bunting, D., Wilkes-Barre, Pa.
Burchard, E. F., Washington, D. C.
Carpenter, R. C., New York, N. Y.
Chase, F. M., Wilkes-Barre, Pa.
Chase, Mrs. F. M., Wilkes-Barre, Pa.
Conner, E. T., Scranton, Pa.
Conner, Mrs. E. T., Scranton, Pa.
Coryell, T., Lambertville, N. J.
Coryell, Mrs. T., Lambertville, N. J.
Coxe, E B., Jr., Drifton, Pa.
Crane, W. R., State College, Pa.
Crichton, A. B., Johnstown, Pa.
Cunningham, J. S., Johnstown, Pa.
Daniels, J., South Bethlehem, Pa.
Darton, N H., Washington, D. C.
Darton, Mrs. N. H., Washington, D. C.
Davis, A. D., Wilkes-Barre, Pa.
Davis, H. G., Kingston, Pa.
Derr, A. F , Wilkes-Barre, Pa.
D'Invilliers, E. V., Philadelphia, Pa.
Dodge, J. M., Philadelphia, Pa.
Dodge, W. F., Wilkes-Barre, Pa.
Dodge, Miss, Wilkes-Barre, Pa.
Dorrance, C., Lansford, Pa.
Drinker, Dr. H. S , S. Bethlehem, Pa.
Dwight, A. S., New York, N. Y.
Dwight, E. W., Philadelphia, Pa.
Edgar, E. R., Wilkes-Barre, Pa.
Emmerich, L. O., Hazleton, Pa.
Emmerich, Mrs. L. O., Hazleton, Pa.
Enzian, C., Wilkes-Barre, Pa.
Eynon, T. N., Philadelphia, Pa.
Eynon, Mrs. T. N., Philadelphia, Pa.
Eynon, Miss, Philadelphia, Pa.
Fackenthal, B. F., Jr , Riegelsville, Pa.
Fernow, B. E., Toronto, Canada.
Firmstone, F., Easton, Pa.
Foote, F. S., Urbana, Ill.
Foster, R. J., Scranton, Pa.
Gleason, F. A., Scranton, Pa.
Gough, H. R., Scranton, Pa.
Gresham, A. L., New York, N. Y.
Griffith, W., Scranton, Pa.
Haddock, J. G., Wilkes-Barre, Pa.
Haldeman, G. T., Wilkes-Barre, Pa.
Hall, H. R., Catasauqua, Pa.
Hall, Mrs. H. R., Catasauqua, Pa.
Hamilton, S. H., New York, N. Y.
Handy, A , New York, N. Y.
Hansell, N. V., New York, N. Y.
Hibbard, H. D., Plainfield, N. J.
Hodge, J. M., Big Stone Gap, Va.
Holbrook, L , New York, N. Y.
Holmes, Dr. J. A., Washington, D. C.
Hood, O. P., Houghton, Mich.
Iredell, F. W., New York, N. Y.
Iredell, Mrs. F. W., New York, N. Y.
Jessup, A. B., Wilkes-Barre, Pa.
Johnson, R. W., Wilkes-Barre, Pa.
Kellam, G. T., Wilkes-Barre, Pa.
Kelly, W , Vulcan, Mich.
King, P. S., Philadelphia, Pa.
Kirchhoff, C., New York, N. Y.
LaMonte, A. C , Scranton, Pa.
Lane, J. S., New York, N. Y.
Lathrop, W. A., Wilkes-Barre, Pa.
Lathrop, Mrs. W. A., Wilkes-Barre, Pa.
Law, A. F., Scranton, Pa.

Ledoux, A. R., New York, N. Y.
Lee, George F., Wilkes-Barre, Pa.
Lentz, L. F., Jr, Mauch Chunk, Pa.
Lentz, W. O., Mauch Chunk, Pa.
Lilly, J., Lambertville, N. J.
Lincoln, J. J., Elkhorn, W. Va.
Lincoln, Mrs. J. J., Elkhorn, W. Va.
Linville, C P., State College, Pa.
Linville, Mrs. C. P., State College, Pa.
Little, J E., Steelton, Pa.
Lloyd, John, Wilkes-Barre, Pa.
Loomis, W. H., Jeddo, Pa.
Ludlow, E, Eccles, W. Va.
Ludlow, Mrs. E, Eccles, W. Va.
Lyle, D. A., St. Daniels, Pa.
Lyle, Mrs. D. A., St. Daniels, Pa.
McMahon, F. J., Wilkes-Barre, Pa.
McMahon, Mrs. F. J, Wilkes-Barre, Pa
Merriman, Mansfield, New York, N. Y.
Merriman, Mrs. M., New York, N. Y.
Merriman, Miss, New York, N. Y.
Miller, B. L., Bethlehem, Pa.
Nicholson, S. T., Wilkes-Barre, Pa.
Norris, R V., Wilkes Barre, Pa.
Norris, Mrs. R. V., Wilkes-Barre, Pa.
Oliver, Gen. P. A., Oliver's Mills, Pa.
Olson, G. L., Ironwood, Mich.
Ormrod, G., Allentown, Pa.
Ormrod, J. D, Emaus, Pa.
Owens, W. D., Pittston, Pa.
Page, G. S., Pittsburg, Pa.
Pardee, I. P., Hazleton, Pa.
Parker, E. W., Washington, D. C.
Pauly, K. A., Schenectady, N. Y.
Perin, C. P., New York, N. Y.
Perin, Mrs. C. P., New York, N. Y.
Pettebone, E. R., Wilkes-Barre, Pa.
Pfordte, O. F., Rutherford, N. J.
Piez, Charles, Chicago, Ill.
Pitman, S. M., Providence, R. I.
Pitman, Mrs. S. M., Providence, R. I.
Pitman, Miss, Providence, R. I.
Prideaux, J. H, Wilkes-Barre, Pa.
Rand, C. F., New York, N. Y.
Raymond, Dr. R. W., New York, N. Y.
Rice, G. S., Pittsburg, Pa.
Richards, J. W., South Bethlehem, Pa.
Richards, W. B., Lansford, Pa.
Richards, W. J., Pottsville, Pa.
Richards, Mrs. W. J., Pottsville, Pa.
Richards, Miss, Pottsville, Pa.
Rushmore, D. B., Schenectady, N. Y.
Saward, F. A., New York, N. Y.
Sharpe, Richard, Wilkes-Barre, Pa.
Sherrerd, A. H., Scranton, Pa.
Sherrerd, Mrs A. H., Scranton, Pa.
Sherrerd, J. M, Easton, Pa.
Sherrerd, Mrs J. M., Easton, Pa.
Shipman, E. H., Bethlehem, Pa.
Smith, J. H, Bridgeport, N. J.
Smith, O, Bridgeport, N. J.
Snyder, B, Jr., Lansford, Pa.
Snyder, Mrs B., Jr., Lansford, Pa.
Souder, H., Cornwall, Pa.
Souder, Mrs. H, Cornwall, Pa.
Spilsbury, E. G, New York, N. Y.
Spilsbury, Miss, New York, N. Y.
Stark, F. M., Wilkes-Barre, Pa
Stark, J W., Wilkes-Barre, Pa.
Stearns, I A., Wilkes-Barre, Pa.
Sterling, P., Wilkes-Barre, Pa.
Sterling, Miss, Wilkes-Barre, Pa.
Stevenson, G. E., Scranton, Pa.
Stewart, Dr. W. S., Wilkes-Barre, Pa.
Stiles, M. D. S., Philadelphia, Pa
Stoek, H. H, Urbana, Ill.
Storrs, A. H., Scranton, Pa.
Storrs, Mrs. A. H, Scranton, Pa.
Straw, C. A., Lansford, Pa.
Struthers, Dr. J., New York, N. Y.
Taylor, K., High Bridge, N. J.
Taylor, Mrs. K., High Bridge, N. J.
Taylor, S. A., Pittsburg, Pa.
Tench, S. F., Lansford, Pa.
Thomas, T., Wilkes-Barre, Pa.
Vallat, B. W., Ironwood, Mich.
Wagner, E B., Wilkes-Barre, Pa.
Warriner, S. D., Wilkes-Barre, Pa.
Warriner, Mrs. S. D., Wilkes-Barre, Pa.
Webb, H. S., Scranton, Pa
Welles, T. L., Wilkes-Barre, Pa.
Whildin, W. G., Lansford, Pa.
Whildin, Mrs. W. G., Lansford, Pa.
Whitaker, F., Allentown, Pa.
Wilbur, W. A., Philadelphia, Pa.
Wilson, E B., Scranton, Pa.
Wilson, Mrs. E. B., Scranton, Pa.
Woodbury, F. E., Milwaukee, Wis.
Woodworth, R. B., Pittsburgh, Pa.
Zerbey, F. E., Wilkes-Barre, Pa.
Zerbey, Mrs. F. E., Wilkes-Barre, Pa.
Zerbey, Miss, Wilkes-Barre, Pa.

Proceedings of the One Hundred and First Meeting, San Francisco, October, 1911.

GENERAL COMMITTEES.

SAN FRANCISCO.—EXECUTIVE, Hon. William C. Ralston, *Chairman;* RECEPTION, Prof. Samuel B. Christy, *Chairman;* SESSIONS, Frederic W. Bradley, *Chairman*. PRESS, H. Foster Bain, *Chairman;* FINANCE, Mark L. Requa, *Chairman;* EXCURSIONS AND ENTERTAINMENTS, Edward H. Benjamin, *Chairman.* Assisted by Harry P. Stow, C. W. Merrill, F. W. Griffin, Gelasio Caetani, Albert Burch, Newton Cleaveland, Corey C. Brayton, and R. E. Cranston.

LOS ANGELES:—EXECUTIVE, Theo. B. Comstock, *Chairman;* R. W. Hadden, *Secretary;* H. R. Simpson, *Treasurer.*

Institute Headquarters at Hotel St. Francis.

The first and opening session, held Tuesday afternoon, Oct. 10, in the Reception-Hall of the St. Francis, was called to order by State Senator William C. Ralston, Chairman of the Executive Committee, who, in a few well-chosen words, welcomed the visiting members and guests of the Institute to San Francisco. Capt. Robert W. Hunt, twice past President of the Institute, and present Acting President for the San Francisco meeting and the subsequent visit to Japan, responded cordially to Mr. Ralston's welcoming address.

By unanimous vote, the Secretary was instructed to send a telegram to President Charles Kirchhoff, expressing regret for his absence, and hoping for a rapid improvement in the health of his mother.

The following papers were presented in brief oral abstract by the authors:

* Electrolytic Refining at the U. S. Mint, San Francisco, Cal., by Edward B. Durham, San Francisco, Cal.

The Parral-Tank System of Slime-Agitation, by Bernard MacDonald, Guanajuato, Mexico.

* The Newport Iron-Mine, Ironwood, Mich., by B. W. Vallat, Ironwood, Mich. (illustrated by lantern-slides).

The Electro-Deposition of Gold and Silver from Cyanide

* Distributed in pamphlet form.

Solutions, by Prof. Samuel B. Christy, Berkeley, Cal.[1] (illustrated by lantern-slides).

During the session, the Secretary read the following telegram from President Kirchhoff:

The committee appointed to consider the best method of perpetuating the name of Samuel Franklin Emmons, late of the United States Geological Survey, have decided that the memorial to him shall take the shape of a research fellowship to be known as the Samuel Franklin Emmons Research Fellowship of Economic Geology. The fellowship is to be administered by Professor Kemp, of Columbia University. Subscriptions are invited by his friends to this fund, which the Committee have fixed at $25,000. Members of the Institute who desire to contribute to the fund will please communicate with the Treasurer, Benjamin B. Lawrence, 60 Wall Street, New York. The Committee consists of George Otis Smith, H. L. Smyth, James Douglas, Joseph A. Holmes, James F. Kemp, F. W. Bradley, J. Parke Channing, Seeley W. Mudd, D W. Brunton, H. Foster Bain, T. A. Rickard, and B. B. Lawrence.

The second session, held Wednesday morning, Oct. 11, in the same place, was called to order by President Hunt, who proffered the chair to Vice-President Gardner F. Williams, of Washington, D. C., and asked him to preside.

The following papers were presented in brief oral abstract by the authors:

Present Conditions in the California Oil-Fields, by Mark L. Requa, San Francisco, Cal.

Present-Day Problems in California Gold-Dredging, by Charles Janin, San Francisco, Cal. (Due to the absence of the author, this paper was presented by Francis J. Dennis, who aided Mr. Janin in its preparation.)

Gold-Production in California, by Charles G. Yale, San Francisco, Cal.

* Mineral Production and Resources of China, by Thomas T. Read, San Francisco, Cal. (illustrated by lantern-slides).[2]

During the session, the Hon. John A. Britton, representing the Panama-Pacific International Exposition, addressed the audience. Later, by unanimous vote, the Secretary was instructed to send a telegram to Arthur D. Foote, of the North Star Mines Co., Grass Valley, Cal., an old and valued member of the Institute, expressing the sincere hope of all present for his rapid recovery from his recent surgical operation.

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* Distributed in pamphlet form.

[1] Not furnished for publication.

[2] *Bulletin* No. 63, Mar., 1912, pp. 293 to 343. Held for vol. xliii.

The third session, held Thursday morning, Oct. 12, at the same place, was called to order by President Hunt, who later asked Dr. R. W. Raymond, Secretary Emeritus of the Institute, to preside.

The following papers were presented in brief oral abstract by the authors:

* The Fritz Engineering and the Coxe Mining Laboratories of Lehigh University, by Joseph Daniels, South Bethlehem, Pa.

* Slime-Filtration, by George J. Young, Reno, Nev.

Coal-Resources of Alaska, by H. Foster Bain, San Francisco, Cal. (Discussed by J. W. Malcolmson, E. W. Parker, and R. W. Raymond.) [3]

During the session, Reiji Kanda, of the Tokyo Institute of Mining, who had just arrived from Japan as the official representative of the Reception Committees in Japan, was introduced by President Hunt. Mr. Kanda brought cordial greetings to the members and guests of the Institute, especially those who will visit Japan.

The fourth and concluding session, held Thursday afternoon, Oct. 12, in the impressive Greek Theater of the University of California, at Berkeley, was called to order by President Hunt, who asked Vice-President S. B. Christy to preside. Professor Christy called attention to the Biographical Notice of Samuel Franklin Emmons, published in *Bulletin* No. 57, September, 1911, and as a friend of long standing he added a few interesting reminiscences from his early personal associations with Dr. Emmons.

Dr. R. W. Raymond, Secretary Emeritus, then presented the second section of his paper, Reminiscences of the Beginning of the Institute.[3] (The first section of this paper was presented at the Wilkes-Barre meeting, March, 1910.)

Mr. George Otis Smith, Director of the U. S. Geological Survey, addressed the members and guests, setting forth the cordial relations and hearty co-operation that have long existed between the Survey and the Institute.

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* Distributed in pamphlet form.

[3] Not furnished for publication.

The following papers were read by title for future publication by the Institute:

* Cyanide-Plant at the Treadwell Mines, Alaska, by W. P. Lass, Treadwell, Alaska.

* The Mining Industry in Japan, by K. Nishio, Tokyo, Japan.[4]

† The Laramie Tunnel, by David W. Brunton, Denver, Colo.[5]

* Notes on the Liberty Bell Mine, by Charles A. Chase, Denver, Colo.

† The Laws of Igneous Emanation, by Blamey Stevens, New York, N. Y.[6]

† Physical Data of Igneous Emanation, by Blamey Stevens, New York, N. Y.[6]

* Electrolytic Oxygen in Cyanide Solutions, by T. H. Aldrich, Birmingham, Ala.

Fuel-Problems of the Pacific, by Oscar H. Reinholt, Pittsburg, Pa.[7]

Government Coal-Mines in the Philippines, by Oscar H. Reinholt, Pittsburg, Pa.[7]

Some Features of Replacement Ore-Bodies, and the Criteria by Means of Which They May be Discovered, by John D. Irving, New Haven, Conn.[7]

* A Modification of the "Gay Lussac" Method for Silver-Bullion Containing Tin, by Luis E. Salas, New York, N. Y.[8]

† Geology of Some Mines in the South of Colombia, S. A., by F. P. Gamba, Tuquerres, Colombia, S. A.[9]

* The Geology of the Tonopah Mining-District, by Augustus Locke, Goldfield, Nev.[10]

* Rapid Estimation of Available Calcium Oxide in Lime Used in the Cyanide Process, by L. W. Bahney, New Haven, Conn.

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* Distributed in pamphlet form.

† Manuscript available for consultation and discussion.

[4] *Bulletin* No. 61, January, 1912, pp. 103 to 147. Held for vol. xliii.

[5] *Idem*, No. 64, April, 1912, pp. 357 to 376. Held for vol. xliii.

[6] *Idem*, No. 64, April, 1912, pp. 411 to 438. Held for vol. xliii.

[7] Not furnished for publication.

[8] *Bulletin* No. 63, March, 1912, pp. 267 to 278. Held for vol. xliii.

[9] Held for vol. xliii.

[10] *Bulletin* No. 62, February, 1912, pp. 217 to 226. Held for vol. xliii.

* Phosphorus in Coking-Coal, by Charles Catlett, Staunton, Va.

Electrical Practice in Mines, by Burton McCollum, Sturgeon Falls, Ontario, Canada.[11]

† The Bearing of the Theories of the Origin of Magnetic Iron-Ores on Their Possible Extent, by Frank L. Nason, West Haven, Conn.[12]

Cyanide Practice at the Santa Gertrudis Mine, Pachuca, Hidalgo, Mexico, by Hugh Rose, Pachuca, Hidalgo, Mexico.[11]

The Black Mountain Coal-District, Kentucky, by J. B. Dilworth, Philadelphia, Pa.[13]

The Flow of Pulverulent Ore Through Orifices, by Ernest A. Hersam, Berkeley, Cal.[11]

* Examination of Dredging-Properties, by Francis J. Dennis, San Francisco, Cal.

† Discussion of J. B. Dilworth's paper, A Method of Calculating Sinking-Funds, and a Table of Values for Ordinary Periods and Rates of Interest, by John Langton.

Excursions and Entertainments.

An account of the train-trip to the Grand Canyon and through southern California, preceding the San Francisco meeting, and the entertainments and excursions in and around San Francisco in connection with the meeting, was printed in *Bulletin* No. 59, November, 1911, pp. v. to xxxviii. A description of the subsequent visit to Japan, and the entertainments in connection therewith, appeared in *Bulletin* No. 61, January, 1912, pp. 1 to 102.

List of Members and Guests (doubtless incomplete) Registered at the San Francisco Headquarters.

Adams, Miss R. A., Orange, N. J.
Atwater, R. M., Jr., New York, N. Y.
Ayres, Mrs. E. L. C., Bound Brook, N.J.
Ayres, W. S., Hazelton, Pa.
Ayres, Mrs. W. S., Hazelton, Pa.
Bain, H. F., San Francisco, Cal.
Bain, Mrs. H. F., San Francisco, Cal.
Beall, A. S. E., San Diego, Cal.
Bellinger, H. P., Syracuse, N. Y.
Bellinger, Mrs. H. P., Syracuse, N. Y.
Benjamin, Edw. H., San Francisco, Cal.
Benjamin, Mrs. Edward H., San Francisco, Cal.
Benjamin, Miss E., San Francisco, Cal.
Berger, George B., Pittsburg, Pa.
Berger, Mrs. R. B., Pittsburg, Pa.

* Manuscript available for consultation and discussion.

† Distributed in pamphlet form.

[11] Not furnished for publication.

[12] Held for vol. xliii.

[13] *Bulletin* No. 62, February, 1912, pp. 149 to 176. Held for vol. xliii.

Boalt, Mrs John H. San Francisco, Cal.
Boyd, Harold E., Milpitas, Cal.
Bradford, S. K., Palo Alto, Cal
Bradley, F. W., San Francisco, Cal.
Bretherton, S. E., San Francisco, Cal.
Brunton, David W., Denver, Colo.
Bryce, Robert A , Cobalt, Canada.
Burch, Albert, San Francisco, Cal.
Busset, A. P., Jr , Campo Seco, Cal.
Cheney, Samuel, San Francisco, Cal.
Christy, S. B , Berkeley, Cal.
Clark, W. B., Baltimore, Md.
Clark, Mrs. W B., Baltimore, Md.
Cleaveland, N., San Francisco, Cal.
Cottrell, F. G., San Francisco, Cal.
Coyne, Miss B. S., Philadelphia, Pa.
Crawford, J. J., San Francisco, Cal.
Cullum, J. Barlow, Pottsville, Pa
Cunningham, E. S., Wonder, Nev.
Dakin, Fred. H , Jr., Berkeley, Cal.
Dakin, Mrs. F H., Jr., Berkeley, Cal.
Daniels, F. H., Worcester, Mass.
Daniels, J., South Bethlehem, Pa.
Davidson, G. W., Chicago, Ill.
Davidson, Mrs. G. W., Chicago, Ill.
Davis, L W., Carbondale, Wash.
Davis, W. J., Jr., San Francisco, Cal.
DeKalb, Courtenay, Tucson, Ariz.
Dennis, Francis J , San Francisco, Cal.
Dickson, Mrs. C. C., New York, N. Y.
Dietrich, W. F., San Francisco, Cal.
Drake, Francis, London, England.
Dumble, E. T., Houston, Texas.
Durham, Edward B., Berkeley, Cal.
Durham, Mrs. E. B , Berkeley, Cal.
Dutton, Charles E., Goldfield, Nev.
Eaton, H. L., Marble, Nev.
Eddy, L. H., New York, N. Y.
Engelhardt, E. N., Oakland, Cal
Farish, John B., Denver, Colo.
Farnum, Herbert C., Bessemer, Ont.
Folsom, D. M., Palo Alto, Cal.
Forbes, D. L. H., El Tigre, Sinaloa, Mexico.
Foucar, E. L., San Francisco, Cal.
Foucar, Mrs. E. L., San Francisco, Cal.
Garrey, G. H , San Francisco, Cal.
Garthwaite, E. H., Oakland, Cal.
Garvin, J. M., Rock Run, Ala.
Gillette, Miss M., Westfield, Mass.
Goodale, Charles W., Butte, Mont.
Goodloe, Meade, Long Beach, Cal.
Griffin, F. P., San Francisco, Cal.
Grunsky, C E , San Francisco, Cal.
Hall, M. L., Palo Alto, Cal.
Hamilton, W. R., San Francisco, Cal.
Hamilton, Mrs W. R , San Francisco, Cal.
Hanks, A. A., San Francisco, Cal.
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PAPERS.

The Agency of Manganese in the Superficial Alteration and Secondary Enrichment of Gold-Deposits in the United States.

BY WILLIAM H. EMMONS,* CHICAGO, ILL

(Canal Zone Meeting, November. 1910)

CONTENTS.

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I. Introduction and Summary.

Ferric iron, cupric copper, and manganitic manganese are present in many mineral waters, and under certain conditions any one of them will liberate chlorine from sodium chloride in acid solutions. Nascent chlorine dissolves gold. Each of these compounds will thus release chlorine at high temperatures, and at low temperatures in concentrated solutions. In cold, dilute solutions, ferric iron will not give nascent chlorine in appreciable quantity in 34 days, and cupric copper is probably even less efficient; but manganitic compounds (supplied by pyrolusite, etc.) liberate chlorine very readily. In a cold solution containing only 1,418 parts of chlorine per million, considerable gold is dissolved in 14 days when manganese is present. It should be expected, then, that those auriferous deposits, the gangues of which contain manganese, would show the effects of the solution and migration of gold more clearly than non-manganiferous ores.

Gold thus dissolved is precipitated by ferrous sulphate. It is, therefore, natural to suppose that gold in such solutions could not migrate far through rocks containing pyrite, since it would be quickly precipitated by the ferrous sulphate produced through the action of air, oxidizing waters, or the gold-solution itself, upon the pyrite. But the dioxide and higher oxides of manganese react immediately upon ferrous sulphate, converting it to ferric sulphate, which is not a precipitant of gold. Consequently, manganese is not only favorable to the solution of gold in cold, dilute mineral waters, but it also inhibits the precipitating action of ferrous salts, and thus permits the gold to travel further before final deposition.

These statements apply to the action of surface-waters descending through the upper parts of an auriferous ore-deposit, since such waters are cold, dilute, acid (*i. e.*, oxidizing) solutions. In deeper zones, where they attack other minerals, they lose acidity, until the manganese compounds, stable under oxidizing conditions, are precipitated together with the gold. Thus, manganite, as well as limonite and kaolin, is frequently found in secondary (*i. e.*, dissolved and reprecipitated) gold-ores. Moreover, in the precipitation of secondary copper and silver sulphides, ferrous sulphate is generally formed; and, consequently, the secondary silver or copper sulphides frequently contain gold.

Those deposits in the United States in which a secondary enrichment in gold is believed to have taken place are, almost without exception, manganiferous. Since secondary enrichment is produced by the downward migration, instead of the superficial removal and accumulation, of the gold, it should follow that both gold-placers and outcrops rich in gold would be found more extensively in connection with non-manganiferous deposits; and this inference is believed to be confirmed by field-observations.

The problem is not as simple as this preliminary statement of it may seem to indicate. Some of the numerous and complex data bearing upon it are collated and discussed in the pages that follow.

Among the papers which treat the superficial alteration and secondary enrichment of copper-, gold-, and silver-deposits, are those of S. F. Emmons,[1] Weed,[2] Penrose,[3] Winchell,[4] Van Hise,[5] Kemp,[6] and Rickard.[7] The processes upon which the changes depend are clearly outlined in these, and subsequent work has, in a large measure, confirmed the premises stated. The chemical laws and physical conditions controlling secondary enrichment have been reviewed in several reports more recently published and examples illustrating the processes have been multiplied. The papers of Lindgren, Ransome, Spencer, Boutwell, Irving, Graton, McCaskey, Spurr, and Garrey and Ball are particularly valuable. Such work has shown that the secondary enrichment of pyritic copper-deposits is an important and almost universal process; that many silver-deposits are enriched by superficial agencies; but that many gold-deposits do not show deep-seated secondary enrichment.

T. A. Rickard[8] has brought out clearly the processes by

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[1] The Secondary Enrichment of Ore-Deposits, *Trans.*, xxx., 177 to 217 (1900).

[2] The Enrichment of Gold and Silver Veins, *Trans.*, xxx., 424 to 448 (1900).

[3] The Superficial Alteration of Ore-Deposits, *Journal of Geology*, vol. ii., No. 3, pp. 288 to 317 (Apr.–May, 1904).

[4] *Bulletin of the Geological Society of America*, vol. xiv., pp. 269 to 276 (1902).

[5] Some Principles Controlling the Deposition of Ores, *Trans.*, xxx., 27 to 177 (1900).

[6] Secondary Enrichment in Ore-Deposits of Copper, *Economic Geology*, vol. i., No. 1, pp. 11 to 25 (Oct.–Nov., 1905).

[7] The Formation of Bonanzas in the Upper Portions of Gold-Veins., *Trans.*, xxxi., 198 to 220 (1901).

[8] *Loc. cit.*

which gold-deposits may be enriched relatively near the surface in the oxidized zone by the removal of valueless minerals which are more readily dissolved than gold. On the problem of deeper-seated precipitation of gold below the zone of oxidation there is less evidence. In some mines, however, the transportation and deep-seated precipitation of gold is clearly shown, as was pointed out long ago by Weed.

While engaged in the investigation of certain auriferous deposits in the Philipsburg quadrangle, Montana, for the U. S. Geological Survey, I was confronted by evidence gained in two important mines, which seemed to be conflicting on this point. In one of them, the Cable mine, there was no evidence that gold had been concentrated by cold solutions below the zone of oxidation, but in the Granite-Bimetallic lode there was enrichment of both gold and silver below the zone of leached oxides. The richer silver-minerals occur in cracks and in small fissures cutting across the banding of the primary deposits and are related very distinctly to the present topography of the country. The evidence therefore appeared to be conclusive that these minerals were deposited by cold mineral waters and that their metallic contents had been dissolved from portions of the lode higher up. The enriched silver-ore carries considerably more gold than the primary ore in the bottom of the mine, and more than the upper portion of the oxidized zone, including the outcrop. No placers have been formed from this deposit, although it has produced considerable gold. On the other hand, important placers have been developed just below the outcrop at the Cable mine. Clearly there has been a kind of selection in the operation of the processes of solution and precipitation of gold.

Although the ores of the two deposits differ in other respects, the most striking difference is in the manganese-content. The abundance of manganese in the Granite-Bimetallic manifests itself in the characteristic coloration of the ores—pink in the unoxidized, brown or black in the oxidized zone. In the Cable, manganese is practically absent. The difference in manganese-content is so striking as to suggest a causal relationship with the equally-marked difference in the amount of secondary enrichment.

The use of manganese in the chlorination process to give

free chlorine, which dissolves gold, is well known. Le Conte[9] said as early as 1879 that free chlorine is the most important natural solvent of gold, and Richard Pearce, in his presidential address before the Colorado Scientific Society, in 1885, recorded experiments in which gold had been dissolved in hot sulphate solutions with common salt and manganese dioxide.[10] Don obtained similar results with more dilute solutions.[11] It appeared desirable, therefore, to ascertain whether these reactions are carried on appreciably in cold dilute solutions similar to mine-waters; and Nicholas Sankowsky and Clarence Russell, in a seminar on the Chemistry of Ore-Deposits, which I conducted at the University of Chicago, compiled all available analyses of waters from gold- and silver-mines in non-calcareous rocks. A. D. Brokaw conducted a series of experiments at my request, using cold dilute solutions of compositions suggested by the analyses. He performed other experiments also, showing the action of manganese dioxide on ferrous salts, which are applicable to the study of the precipitation of gold. During the progress of this investigation, W. J. McCaughey, of the Bureau of the Mint, Washington, D. C., published his valuable paper on the solvent effect of ferric and cupric salt solutions upon gold,[12] and this in a large measure supplemented the work carried on in the seminars at the University of Chicago.

The experiments conducted by Brokaw showed that manganese in the presence of chlorides and sulphates is very much more efficient in the reactions dissolving gold than are the other salts which are common in mine-waters. To verify these results by field-evidence, the review of the literature was taken up in greater detail, and there also the results indicate a marked difference in the behavior of the cold dilute mineral waters in the presence and in the absence of manganese. Lindgren's classification of the gold-deposits of North America has been of great value in reviewing these deposits; since in the United States manganese is rarely a gangue-mineral in the primary gold-deposits as old as the early Cretaceous California gold-veins, whereas it is frequently

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[9] *Elements of Geology*, p. 285.

[10] *Proceedings of the Colorado Scientific Society*, vol. ii., p. 3 (1885–87).

[11] *Trans.*, xxvii, 654 (1897).

[12] *Journal of the American Chemical Society*, vol. xxxi., No. 12, pp. 1261 to 1270 (Dec., 1909).

present in very appreciable quantities in those deposits which were formed nearer the surface and which are related to intrusives of Tertiary age. Possibly this difference is due to conditions of temperature and pressure which prevailed when the deposits were formed.[13] Since there are no data which show the effect of highly-carbonated waters on these reactions, I have as far as possible eliminated examples of gold-deposits in limestone, and the discussion is confined mainly to deposits in non-calcareous rocks. I have not attempted to review exhaustively the evidence afforded by deposits outside of the United States with respect to the hypothesis suggested, but some of these deposits appear to supply accurate confimatory data.

In a statistical study of outcrops, to ascertain whether gold is more extensively leached in manganiferous lodes than in the outcrops of those which do not carry manganese, and whether placers are more frequently developed in connection with non-maganiferous lodes, the reports of Dr. R. W. Raymond,[14] written soon after the discoveries of many of the deposits, have been of great value.

I wish to acknowledge my indebtedness to my colleagues of the U. S. Geological Survey, and to many other geologists whose accurate observations I have drawn upon to test the hypothesis. Their conclusions respecting the secondary enrichment of gold appear to support the hypothesis and, differing as they do with respect to the migration of gold in particular deposits, they become reconciled when inspected from this view-point, and thus they are themselves supported. Dr. R. C. Wells, of the U. S. Geological Survey, has read critically certain portions of this paper, where the principles of physical chemistry are involved.

II. Salts Contained in the Waters of Gold- and Silver-Mines in Non-Calcareous Rocks.

The composition of mine-waters depends upon the character of the ore and wall-rock and the position of the deposit with respect to bodies of salt water. There are certain compounds which are generally present, and some which nearly always predominate. Of the few analyses which have been made of

[13] W. Lindgren, The Relation of Ore-Deposition to Physical Conditions, *Economic Geology*, vol. ii., No. 2, pp. 105 to 127 (Mar.–Apr., 1907).

[14] *Mines and Mining West of the Rocky Mountains* (1868–1875).

waters from gold-mines, a large proportion are incomplete; and it is not always stated whether compounds not reported were looked for. Sankowsky and Russell, utilizing all data available to them, recalculated the analyses to the ionic form of statement, and where necessary to parts per million, and made a general average of the results. Where compounds were not reported in the analyses it was assumed that they were not present. Arsenic, antimony, and other elements, small traces of which must be present in some waters, are not reported. Since the averages were obtained by dividing the sums by the total number of analyses (29) and not by the number of analyses showing a particular element, and since some analyses are incomplete, any corrections applied for this source of error would tend to increase the number of parts per million indicated. On the other hand, some of the mine-waters were taken from places protected from the more active vadose circulation, and are clearly more concentrated than the major part of the waters. The average of analyses, although a rude approximation, is useful, since it gives some quantitative value to their factor in the problem, and indicates the general nature of the cold solutions in which the metals are transported.

TABLE I.—*Average of 29 Analyses of Waters Taken from Gold-, Silver-, and Gold-Silver Mines in Non-Calcareous Rocks.* (*Compiled by N. Sankowsky and C. Russell.*)

	Parts Per Million.	Number of Determinations.	Absent or Not Determined
Cl [a]	873 10	22	7
SO_4	7,292.29	13	16
CO_3	77.59	7	22
NO_3 [a]	0.06	1	28
PO_4	0.00	traces in 2	27
SiO_2	34.94	12	17
K	17.25	7	22
Na [a]	261.20	9	20
Li	0.10	1	28
Ca	295.00	11	18
Sr	0.06	1	28
Mg	242.44	9	20
Al	333.65	6	23
Mn	30.91	6	23
Ni	trace	traces in 3	26
Co	trace	traces in 3	26
Cu.	5.09	2	27
Zn	2.70	5	24
Fe ii	277.66	22	7
Fe iii	603.07	25	4
H (in acids)	97.26	10	19

[a] Probably too high (see discussion).

1. *Sulphates.*

Primary gold-ores generally carry pyrite, which, oxidizing at or near the surface, yields ferrous sulphate, ferric sulphate, and sulphuric acid. The acid is not formed directly from galena, PbS, or from zinc-blende, ZnS, but pyrite, FeS_2, carries more sulphur than is required to supply SO_4 radical to satisfy the iron, even if ferric sulphate, $Fe_2(SO_4)_3$, is formed instead of $FeSO_4$. As lately shown by Buehler and Gottschalk,[15] galena and zinc-blende dissolve very much more slowly in the absence of FeS_2. The reaction probably requires free acid, which the iron sulphide, owing to its excess of sulphur, supplies. The sulphuric acid from pyrite is increased also by the hydrolization of ferric sulphate and the deposition of limonite.

In Table I. the sulphate radical (7,292 parts per million) is nearly ten times as abundant as all other negative ions and is also in excess of bases, so that on any basis of adjustment to form salts much H_2SO_4 remains. The table shows also an average of 97.26 parts per million of hydrogen in acid. In view of the low atomic weight of hydrogen, this indicates the strongly acid character of the solutions.

2. *Chlorides.*

Chlorine is present in most mine-waters. In 22 out of the 29 analyses it is reported as traces or as determined quantities. The average of 29 analyses shows 873 parts per million, but if the one abnormally rich sodium-chloride water of Silver Islet, Lake Superior, is excluded, the remaining 28 analyses show but 111 parts per million. This figure is probably a better average. It would be further reduced some 2 or 3 parts by excluding the waters of the Geyser mine, Silver Cliff, Colo., which may have come from a deep source. With these two exceptions, it is noteworthy that the waters from mines remote from salt water contain less chlorine than those near the sea or in undrained areas. The distribution of chlorine is an important element in the migration of gold, and therefore I shall consider the sources of chlorine in some detail.

The salt in sedimentary rocks may be dissolved by ground-water. From the available analyses it appears that this source is of less importance than would be supposed. The chlorine-

[15] *Economic Geology*, vol. v., No. 1, p. 30 (Jan., 1910).

content of composite samples of 78 shales and of 253 sandstones was only a trace, while an analysis of a composite of 345 limestones showed only 0.02 per cent.[16] A few rock-making minerals, such as chlor-apatite, scapolite, haüyne, and nosean, contain combined chlorine; but of these all but apatite occur mainly in very rare types of rocks. In some rocks chlorine is present probably as NaCl in the solid particles contained in fluid inclusions. The work of R. T. Chamberlin, A. Gautier, and others, has shown that many granular igneous rocks, when heated to high temperatures, give off gases equal to several times their own volume. While further inquiry of this character is desirable, it is probably true that in general but little chlorine is present in such gases. But gases from certain volcanic rocks, such as obsidian, often contain a high proportion of chlorine and chlorides. Albert Brun[17] has shown that some of the Krakatoa lavas contain gases which equal about one-half the volume of the rock, and that more than half of such gases consists of chlorine, hydrochloric acid, and sulphur monochloride.

Apatite, though widespread in igneous rocks, is a very stable mineral, and consequently cannot be looked upon as an important source of chlorine, although it may contribute small amounts when exposed to favorable conditions of weathering. The average chlorine-content of igneous rocks is, according to F. W. Clarke, 0.07 per cent.

Chlorine is present in nearly all natural waters. Its chief source is from finely-divided salt or salt water from the sea and from other bodies of salt water. The salt is carried by the wind and precipitated with rain.[18] The amount of chlorine in natural waters varies with remarkable constancy with the distance from the shore; several determinations very near the seashore show from 10 to 30 parts of chlorine per million; a few miles away it is generally about 6 parts per million; 50 miles from shore it is generally less than 1 part per million. A surface-

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[16] F. W. Clarke, *Bulletin No.* 330, *U. S. Geological Survey*, p. 27 (1908)

[17] Quelques Recherches sur le Volcanisme aux Volcans de Java. Cinquième partie. Le Krakatau. *Archives des Sciences physiques et naturelles*, Genève, vol. xxviii, No. 7 (Juillet, 1909).

[18] D. D. Jackson, The Normal Distribution of Chlorine in the Natural Waters of New York and New England, *Water Supply and Irrigation Paper No.* 144, *U. S. Geological Survey* (1905).

water from a reservoir at Leadville contained 1.14 parts of Cl per million.[19] The isochlores parallel the shore-line with great regularity, as indicated in the map, Fig. 1, taken from Jackson's report. The amount of chlorine contributed from this source even near the seashore appears small (from 6 to 10 parts per million); but it may be further concentrated in the solutions by evaporation or by reactions with silver, lead, etc., forming chlorides, which in the superficial zone may subsequently be changed to other compounds. In arid countries, as suggested by C. R. Keyes, dust containing salt doubtless contributes chlorine to the mine-waters. Penrose,[20] discussing the distribution of the chloride ores, pointed out long ago that these minerals form most abundantly in undrained areas.

3. *Carbonates and Alkaline Earths.*

The analyses in Table I. do not include those from mines in limestones. The carbonate reported gives an average of 77 parts per million. In the acid waters under consideration, the carbonates of the bases would necessarily be present as bicarbonates, although this fact is not indicated in the analyses.

Even in non-calcareous rocks considerable calcium (295 parts per million) and magnesium (242 parts) are carried by the waters. They are derived in part from reactions between the acid sulphates and the silicates of the wall-rock.

4. *Alumina.*

In some waters aluminum sulphate is abundant (the average of aluminum, 333 parts per million). It forms where sulphate waters attack kaolin, setting free SiO_2 and taking alumina into solution. The above average is probably high on account of one concentrated alum-water in a Comstock mine.[21]

5. *Nitrates.*

Nitrates are not abundant in mine-waters. In one analysis only[22] is NO_3 reported (1.60 parts per million), and this in a deep-seated water of questionable genesis.

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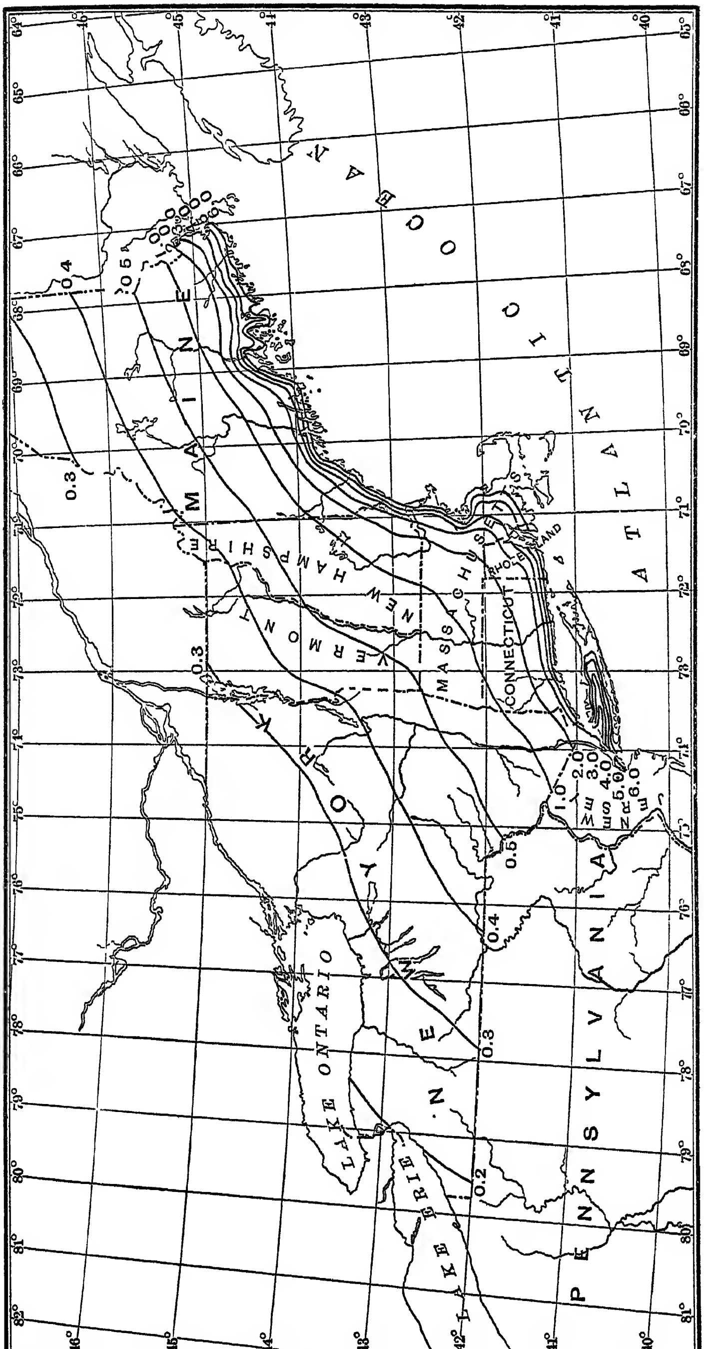
[19] S. F. Emmons, Geology and Mining Industry of Leadville, Colorado, *Monograph No. XII, U. S. Geological Survey*, p. 552 (1886).

[20] *Journal of Geology*, vol. ii., No. 3, p. 314 (April–May, 1894).

[21] *Bulletin of the Department of Geology, University of California*, vol. iv., No. 10, p. 192 (1904–06).

[22] Geyser Mine, Silver Cliff, Colo. See S. F. Emmons, *Seventeenth Annual Report, U. S. Geological Survey*, Part II., p. 462 (1895–96).



The numbers on the isochlores indicate the parts of chlorine per million in the natural waters.

FIG. 1.—NORMAL CHLORINE MAP OF NEW ENGLAND AND NEW YORK.

(Compiled by Daniel D. Jackson, U. S. Geological Survey, *Water Supply and Irrigation Paper No.* 144.)

6. *Phosphates.*

Traces only of PO_4 are reported from two mine-waters; others contained none, if determinations were made.

7. *Silica.*

Silica (35 parts per million) appears high for acid waters. The analyses include a manganiferous sulphate water from the Comstock, abnormally high in silica.[23]

8. *Iron.*

Iron is the most abundant metal in the waters of gold-mines. Ferric iron (603 parts per million) is, according to these analyses, more than twice as abundant as ferrous iron (277 parts per million). Probably too little attention has been given to the state of oxidation of iron in unaltered mine-waters. Ferrous salts in solution, when exposed to air, rapidly become ferric; yet, so far as I know, no mine-water which has clearly not had access to air has been examined with respect to the state of oxidation of the iron. Ferrous iron is much more abundant below than above the water-table.

9. *Manganese.*

If manganiferous minerals are present in the primary ore, they oxidize in the upper portion of the deposit to manganese dioxide or other high oxides of manganese; and these, in turn, oxidize ferrous sulphate, in the presence of sulphuric acid, to ferric sulphate. Consequently, the iron in manganiferous waters is likely to be in the oxidized state.

10. *Copper.*

One analysis shows 147 parts of copper per million. Two other analyses show traces. Small amounts must be present in many other waters, since gold-ores often carry copper. Possibly, small traces of the heavy metals were not looked for in many of the waters analyzed.

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[23] *Bulletin of the Department of Geology, University of California*, vol. iv., No. 10, p. 192 (1904–06).

III. Chemical Experiments in the Solution and Deposition of Gold.

The superficial alteration of gold-deposits and the migration of gold in the deposits take place at low temperatures. At the very surface the temperatures range between 0° and 50° C. and pressures do not exceed one atmosphere. At the normal gradient of increase, the temperatures, even several thousand feet below water-level, would not exceed 100° C., and in the main are considerably lower. The general character and, approximately, the concentration of the solutions are known from the analyses of mine-waters. The conditions are fairly constant. From the mass of chemical data relating to the subject, the following experiments seem to be particularly suggestive in connection with the present problem.

1. Stokes[24] placed gold leaf in a solution containing 25 g. per liter of ferric sulphate, and, after heating to 200° C., found that not a trace of gold had been deposited in the cold part of the sealed tube in which the experiment was carried on. This experiment does not confirm the statement frequently made that ferric sulphate will dissolve gold.

2. Don[25] exposed to air finely-divided gold and auriferous sulphide ores in solutions containing from 1 to 20 g. of ferric chloride and ferric sulphate per liter of water; and after several months no gold had been dissolved. Presumably the gold was not mixed with the sulphide in all of the experiments.

3. W. J. McCaughey,[26] upon boiling for several hours 50 cc. of HCl (sp. gr. 1.178) diluted to 125 cc. with 250 mg. of gold, found there was no loss of gold.

4. In a bent tube Stokes[27] heated gold leaf for 16 hr. at 200° C. in a solution composed of 85 g. of cupric chloride and 133 cc. of 20 per cent. HCl in a liter of water. The gold leaf was dissolved and redeposited in the upper portion of the tube. He writes the reaction as follows:

$$Au + 3\,CuCl_2 \leftrightarrows AuCl_3 + 3\,CuCl.$$

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[24] *Economic Geology*, vol. i., No. 7, p. 650 (July–Aug., 1906).

[25] *Trans.*, xxvii., 598 (1897).

[26] *Journal of the American Chemical Society*, vol. xxxi., No. 12, p. 1263 (Dec., 1909).

[27] *Op. cit.*, vol. i., p. 649.

5. Stokes[28] heated gold leaf to 200° C. in a closed tube containing a solution of 25 g. of ferric sulphate and 0.01 g. of NaCl. Gold was dissolved in 40 hours.

6. Stokes[29] found that at 200° C. gold leaf was dissolved in a mixture of 2 parts of 20 per cent. solution of ferric chloride and 1 part of 20 per cent. solution of HCl.

7. W. J. McCaughey[30] dissolved gold at from 38° to 43° C., in hydrochloric acid solutions of ferric sulphate. The results are indicated by the curves in Fig. 2. Solution *A* contained 1 g. of iron, introduced as ferric sulphate, and 25 cc. of HCl

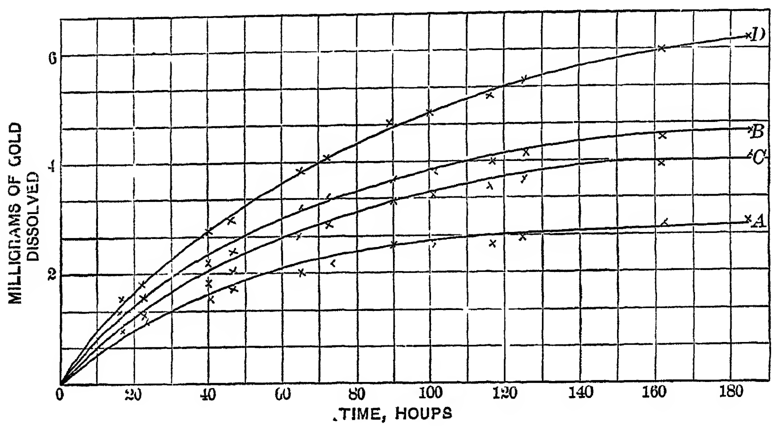

FIG 2.—DIAGRAM SHOWING THE RATE OF SOLUTION OF GOLD IN CONCENTRATED SOLUTIONS OF HYDROCHLORIC ACID AND FERRIC SULPHATE. (Illustrating Experiment No. 7, by McCaughey.)

(sp. gr. 1.178) in a solution diluted to 125 cc. containing 250 mg. of gold rolled to 0.009 in. Solution *B* contained the same amount of iron sulphate and 50 cc. of HCl. Solution *C* contained 2 g. of Fe as ferric sulphate and 25 cc. of HCl. Solution *D* had twice the concentration of *A*. The diagram shows the amount of gold dissolved after different periods of treatment.

8. McCaughey[31] found that gold is dissolved at from 38° to 43° C. in a strong solution of cupric chloride and HCl. The

[28] *Economic Geology*, vol. i., No. 7, p. 650 (July–Aug., 1906).

[29] *Idem*, p. 650.

[30] *Journal of the American Chemical Society*, vol. xxxi., No. 12, p. 1263 (Dec., 1909).

[31] *Idem*, p. 1264.

amounts dissolved are shown by the curves in Fig. 3. Solution *A* contained 1 g. of Cu as cupric chloride and 25 cc. of HCl (sp. gr. 1.178); solution *B*, 1 g. of Cu as $CuCl_2$, and 50 cc. of HCl; solution *C*, 2 g. of Cu as $CuCl_2$ and 25 cc. of HCl; and solution *D*, 2 g. of Cu as $CuCl_2$ and 50 cc. of HCl; the final solution being in all cases diluted to the volume of 125 cc. The diagram shows that *D*, which was twice as concentrated as *A*, dissolved about 12 times as much gold.

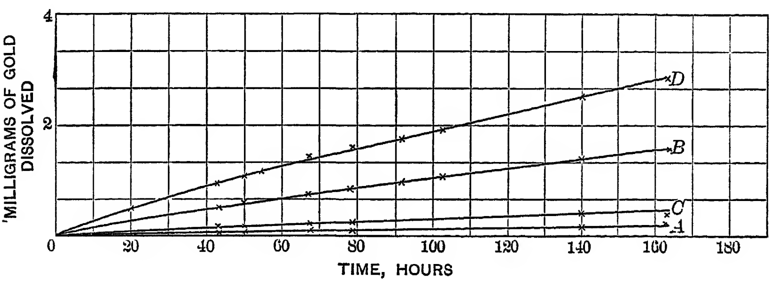

Fig. 3.—Diagram Showing the Solubility of Gold in Concentrated Solutions of Hydrochloric Acid and Cupric Chloride.

(Illustrating Experiment No. 8, by McCaughey.)

9. Richard Pearce[32] placed native gold in a flask containing hydrated manganese dioxide with 40 g. of salt and 5 or 6 drops of H_2SO_4. After heating for 12 hr. appreciable gold had been dissolved.

10. T. A. Rickard[33] extracted 99.9 per cent. of the gold from rich manganiferous ore with a solution of ferric sulphate, common salt, and a little H_2SO_4

11. Don[34] found that 1 part of HCl in 1,250 parts of H_2O, in the presence of MnO_2, dissolves appreciable gold.

A number of experiments on the solubility of gold in cold dilute solutions were made at my request by A. D. Brokaw.[35] The nature of these experiments is shown by the following statements, in which (*a*) and (*b*) represent duplicate tests:

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[32] *Trans.*, xxii., 739 (1893).
[33] *Trans.*, xxvi., 978 (1896).
[34] *Trans.*, xxvii., 599 (1897).
[35] *Journal of Geology*, vol. xviii., No. 4, pp. 321 to 326 (May-June, 1910).

12. $Fe_2(SO_4)_3 + H_2SO_4 + Au$.
(*a*) no weighable loss. (34 days.)
(*b*) no weighable loss.

13. $Fe_2(SO_4)_3 + H_2SO_4 + MnO_2 + Au$.
(*a*) no weighable loss. (34 days.)
(*b*) 0.00017 g. loss.[c]

[c] This duplicate was found to contain a trace of Cl, which probably accounts for the loss.

14. $FeCl_3 + HCl + Au$.
(*a*) no weighable loss. (34 days.)
(*b*) no weighable loss.

15. $FeCl_3 + HCl + MnO_2 + Au$.
(*a*) 0.01640 g. loss. Area of plate, 383 sq. mm. (34 days.)
(*b*) 0.01502 g. loss. Area of plate, 348 sq. mm.

In each experiment the volume of the solution was 50 cc. The solution was one-tenth normal with respect to ferric salt and to acid. In experiments 13 and 15, 1 g. of powdered manganese dioxide was also added. The gold, assaying 999 fine, was rolled to a thickness of about 0.002 in.; cut into pieces of about 350 sq. mm. area, and one piece, weighing about 0.15 g., was used in each duplicate.

To approximate natural waters more closely, a solution was made one-tenth normal as to ferric sulphate and sulphuric acid, and one twenty-fifth normal as to sodium chloride. Then 1 g. of powdered manganese dioxide was added to 50 cc. of the solution, and the experiment was repeated. The time was 14 days.

16*a*. $Fe_2(SO_4)_3 + H_2SO_4 + NaCl + Au$.
No weighable loss.

16*b*. $Fe_2(SO_4)_3 + H_2SO_4 + NaCl + MnO_2 + Au$.
Loss of gold, 0.00505 gram.

The loss is comparable to that found in experiment 15, allowing for the shorter time and the greater dilution of the chloride.

To determine whether the free acid or the ferric chloride is

the solvent, experiment 17 was made, in which 50 cc. of one-tenth normal HCl was used with 1 g. of powdered MnO_2.

17. $HCl + MnO_2 + Au$.
Loss of Au, 0.01369 g. Time, 14 days.

In experiment 18, sodium hydroxide was added to 50 cc. of one-tenth normal ferric chloride solution until the precipitate

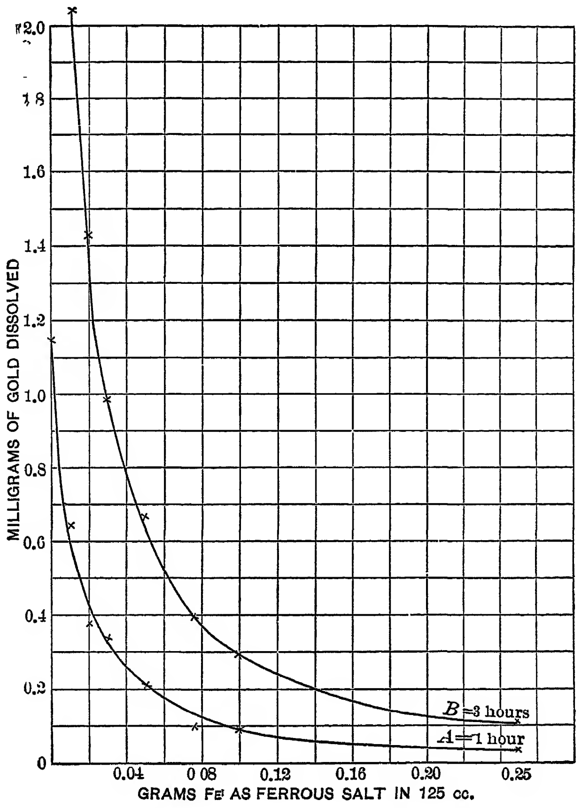

FIG. 4.—DIAGRAM ILLUSTRATING THE EFFECT OF FERROUS SULPHATE IN SUPPRESSING THE SOLUBILITY OF GOLD IN FERRIC SULPHATE SOLUTIONS, WHERE GOLD IS DISSOLVED AS CHLORIDE.

(Illustrating Experiment No. 19, by McCaughey.)

formed barely re-dissolved on shaking, after which 1 g. of powdered MnO_2 was added.

18. $FeCl_3 + MnO_2 + Au$.
Loss of Au, 0.00062 g. Time, 14 days.

These results show, that in the presence of manganese dioxide, free hydrochloric acid is more efficient than ferric chloride. The same amount of chlorine was present in both solutions.[36]

19. McCaughey's experiments show the effect of very small amounts of ferrous sulphate on solutions of gold in ferric sulphate. To a solution, 125 cc., containing 1 g. of iron as ferric sulphate and 25 cc. of HCl, ferrous sulphate was added in quantities containing from 0.01 to 0.25 g. of ferrous iron. The solutions were immersed in boiling water and subsequently 250 mg. of gold was added. The dissolved gold was determined at the end of 1 hr. and 3 hr. At the end of 3 hr. the gold dissolved was greater, probably because some ferrous sulphate had changed to ferric sulphate. Even 0.01 g. of the ferrous iron greatly decreases the solubility of gold in the ferric sulphate and HCl solution, and 0.25 g. of ferrous sulphate drives nearly all the gold out of solution. These experiments are illustrated by Fig. 4, in which the horizontal lines represent ferrous salt put in the mixture and the vertical lines the amount of gold (in milligrams) dissolved by chlorine in the solution. The lower curve represents conditions at the end of 1 hr., the upper curve at the end of 3 hr., when some of the ferrous salt had oxidized by contact with the air.

20. To determine the rate at which ferrous sulphate, in the presence of sulphuric acid and manganese dioxide, would be oxidized to the ferric salt, Brokaw made the following experiment:

100 cc. of 1.6 normal $FeSO_4$ was acidified with sulphuric acid and shaken vigorously with 5 g. of powdered MnO_2. After 5 min., the solution was filtered. No ferrous iron was detected by the ferricyanide test, showing that the iron had been completely oxidized to the ferric state.

IV. Discussion of Experiments.

1. *Nitrates.*

Dilute acid nitrate-chloride waters readily dissolve gold, since they are equivalent to weak aqua regia. The chlorine

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[36] Brokaw, *Journal of Geology*, vol. xviii., No. 4, pp. 322 to 323 (May-June, 1910).

set free by the reaction oxidizing HCl is more active than a solution of chlorine in water, and converts gold into gold chloride. For present purposes we may consider that the reaction is as follows:

$$9\ HCl + 3\ HNO_3 + 2\ Au \rightarrow 6\ H_2O + 3\ NOCl + 2\ AuCl_3.$$

Nitrosyl chloride, NOCl, which is formed in this reaction, does not react directly with gold, but is thought by some to affect the reaction favorably as a catalytic agent. Whether this is true or not, in each of the reactions by which gold is dissolved in chloride solution its solvent power may be ascribed to its "nascent" state. In this reaction, as in those which follow, the presence of an element with more than one valence is a necessary condition, and its valence is reduced as gold passes into solution.

The reaction given above, $3\ HCl + NHO_3$, may be written as follows:[37]

$$Cl - H + H - O - \overset{\overset{\displaystyle O}{\|}}{N} = O + 2H\,.\,Cl \rightarrow 2H_2O + Cl_2 + Cl - N = O.$$

The chlorine reacts with gold, forming soluble gold chloride.

$$2\ Au + 3\ Cl_2 \rightarrow 2\ AuCl_3.$$

With regard to the latter reaction, Dr. R. C. Wells, of the U. S. Geological Survey, supplied the following note:

"The reaction ($2\ Au + 3\ Cl_2 \rightarrow 2\ AuCl_3$) aims to express the initial and final stages, but says nothing of the mechanism of the reaction or the necessity for the chlorine being in the 'nascent' state. In accordance with present theories, a 'nascent' chlorine atom, while taking a negative charge to form chloride, allows the corresponding positive charge to ionize the gold,

$$Au + \odot = Au^{\bullet}.$$

This ionization occurs with greater difficulty in the case of gold than with almost any other metal. The aurous ion passes with great readiness into the auric ion, $Au^{\bullet\bullet\bullet}$. Moreover, both ions form complexes with chlorides. The effectiveness of

[37] Alexander Smith, *General Inorganic Chemistry*, p. 449 (1907).

chlorine in dissolving gold in accordance with this theory may be ascribed partly to the production of the complex gold chloride ions, thus removing the gold ions from solution with such effectiveness that more gold ionizes, and thus the process continues until equilibrium is established."

In the 29 analyses of mine-waters NO_3 is reported from but one (Geyser mine, Silver Cliff, Colo., 1.6 parts per million), and this is a water of questionable genesis. Possibly, nitrates are more abundant than is indicated by the analyses; and if so, they must increase the solvent power of chloride solutions; but the data at present available do not indicate that they affect the superficial reactions to any important extent.

2. *Manganese Oxides.*

That gold is dissolved in moderately dilute solutions containing salt and manganese oxides is shown by experiments 11, 15 and 16. The reaction with manganese used to prepare chlorine commercially is illustrated by the following equation: (The reaction is not so simple as stated. It is discussed later.)

$$Mn^{iiii}O_2 + 2\ NaCl + 3\ H_2SO_4 \rightarrow 2\ H_2O + 2\ NaHSO_4 + Mn^{ii}SO_4 + 2\ Cl.$$

At the beginning of the reaction the manganese has a valence of four; at the end a valence of two. With acid the reaction may be as follows:

$$MnO_2 + 4\ HCl \rightarrow 2\ H_2O + MnCl_2 + Cl_2.$$

Besides the presence of a chloride, some other conditions are essential to the solution of gold. There appear to be two. One is that some other substance must also be present which is capable of being reduced so as to liberate chlorine—as, for example, a ferric salt which may be reduced to the ferrous, a cupric to the cuprous, the higher manganese salts to the lower, etc. The other is the evolution of "nascent" chlorine. This is particularly illustrated by the action of aqua regia or the production of chlorine by hydrochloric acid and pyrolusite. In short, any of a number of methods of producing free chlorine would be effective in the solution of gold. Possibly both of the conditions just mentioned may in the last analysis be identical.

The essential point is that the atomic chlorine in a state of molecular exchange or evolution is able to combine with the gold. For present purposes the gold may be considered to dissolve as gold chloride, although chemical investigations favor the theory that a complex ion containing gold is formed. The only consideration which becomes important in its geological aspect is the presence of the compounds which not only admit of easy changes of valence, but which act upon hydrochloric acid with the production of free chlorine.

In mine-waters chlorine is supplied as NaCl.

$$\begin{array}{llllll} 16b.\ Fe_2\,(SO_4)_3 & + H_2SO_4 & + NaCl & + MnO_2 & + Au \\ \quad N/10 & N/10 & N/25 & 1\text{ g.} & 0.15\text{ g.} \end{array}$$

0.00505 g. loss of gold by solution in 14 days (cold).

Under the same conditions without manganese there was no weighable loss (see experiment 16*a*).

As used herein the normal solution contains 1 g.-equivalent of the solute in 1 l. of solution. A solution normal with respect to chlorine contains 1 g. of chlorine times 35.45, the molecular weight of chlorine, in 1 l. of solution.

In this experiment the concentration of Cl (1,418 parts per million) is not so great as has been observed in a few mine-waters, and not more than three times as great as Don determined in waters from a number of Australasian mines.[38] The solutions, however, contain more chlorine than the average of 29 analyses of mine-waters (873 parts of Cl per million), considerably more than that of 28 analyses (111 parts of Cl per million), and more than most mine-water analyses from American gold-mines.

Manganese is abundant in many gold-bearing deposits; is sparingly represented in some; and from a very large number it has not been reported. The chief primary minerals are the carbonates (rhodochrosite and manganiferous calcite), the silicate (rhodonite), amethystine quartz, and the less-abundant sulphide, alabandite. Some rock-making minerals carry small amounts of manganese. It readily forms sulphates, chlorides, etc., and is dissolved by acid mine-waters. In the 29 analyses of Table I. it is reported from 6 mines. In some waters it is abundant.

[38] *Trans.*, xxvii., 654 (1897).

The average of the 29 shows 30.9 parts per million. Even a little manganese generally stains the gossan black or chocolate-brown, and consequently it is readily recognized in the oxidized ores. Manganese changes its valence more readily than other elements common in gold-ores and it is in many respects unique among the elements. The following note is abridged from Alexander Smith, *General Inorganic Chemistry*, p. 737:

It stands alone on the left side of the eighth column of the periodic table; the right side of that column is occupied by the halogens. It is never univalent as the halogens are; but the heptoxide, Mn_2O_7, and corresponding permanganic acid, $HMnO_4$, are in many ways closely related to the heptoxide of chlorine and perchloric acid, $HClO_3$. Permanganic acid is a very active acid. Contrary to the habit of feebly acidic and feebly basic oxides such as those of zinc, aluminum and tin, the basic oxides of manganese arè not at all acidic and the acidic oxides, with the possible exception of Mn_2O_3, are not also basic. There are thus five rather well defined sets of compounds showing five different valences of the element.

These include manganosite (MnO), pyrochroite ($MnO.H_2O$), manganite ($Mn_2O_3.H_2O$), hausmannite (Mn_3O_4), pyrolusite (MnO_2), psilomelane, etc.

3. *Lead Oxides.*

Lead oxide, like manganese oxide, is said to facilitate the solution of gold[39] when added to solutions of ferric sulphate and sodium chloride. Lead is both bivalent and quadrivalent and forms corresponding oxides and hydroxides. These, however, are generally not abundant in the oxidized zones of lead-bearing ore-deposits, probably because the lead carbonate and the sulphate are relatively insoluble in water and usually are formed instead of the oxides. Lead is reported in but one of the 29 analyses of waters from gold- and silver-mines, tabulated above, and in this case the water carried but 1.35 parts per million. Many gold-deposits contain but little lead and some contain none. It is believed to be of very subordinate importance in connection with the solution of gold.

4. *Ferric Compounds.*

As shown by experiments, gold is not dissolved by hydrochloric acid, by ferric sulphate, or by ferric chloride. It is

[39] Victor Lehner, *Journal of the American Chemical Society*, vol. xxvi., No. 5, p. 552 (May, 1904).

dissolved at 38° C. in concentrated solution containing both ferric sulphate and hydrochloric acid.

5. *The Efficiency of Ferric Iron and Cupric Copper to Supply Nascent Chlorine, Compared with that of Manganitic Manganese.*

As shown by experiment 4, a concentrated solution of $CuCl_2$ with HCl dissolves appreciable gold at 200°, and Fig. 3 shows that a solution containing 1 g. of copper as cupric chloride and 25 cc. of HCl (sp. gr., 1.178) in 125 cc. of solution at 38°+, dissolves 0.23 mg. of gold in 163 hr. Since cupric copper and ferric iron are present in many mineral waters, the nature of these reactions should be considered in some detail in order to compare their efficiency with that of manganitic manganese.

Solutions of ferric sulphate with sulphuric acid and salt dissolve gold at high temperatures. Concentrated solutions of ferric sulphate and hydrochloric acid dissolve gold at from 38° to 43° C. In the cold, the reaction may go on in concentrated solutions, but in those approximating the concentration of mine-waters (and one of them considerably more concentrated than most mine-waters) no weighable loss of gold was obtained. With MnO_2 under the same conditions there was a very appreciable loss in a solution containing only 1.4 g. of Cl in a liter. It appears, therefore, that the action of ferric iron on gold in cold dilute mine-waters with H_2SO_4 and NaCl is probably negligible; for the experiments with ferric iron in such solutions, without manganese, extended over a period of 34 days without weighable loss of gold.

Many auriferous deposits contain copper; and it is desirable to compare the efficiency of cupric with ferric salts and with manganitic salts in similar solutions. Since the reactions which give nascent chlorine are conditioned upon the presence of some element that changes its valence in the reactions, and since the processes underground take place in sulphate solutions, it did not appear necessary, after ferric salt had been shown to be incompetent, to conduct experiments with copper; for, as is well known, cuprous salts, though they may be present, have never been detected in acid sulphate mine-waters, whereas ferric and ferrous sulphate are very common in such waters.

Fig. 2 shows that in 163 hr. a solution carrying 2 g. of ferric iron as sulphate and 50 cc. of HCl (sp. gr. 1.178) diluted to 125 cc., with 250 mg. of gold in the solution, dissolves 6 mg. of gold. In the same time, as shown by Fig. 3, a solution containing 2 g. of copper as cupric chloride and 50 cc. of HCl diluted to 125 cc., dissolves but 2.84 mg. of gold, the same amount of gold being exposed. These results indicate that, in concentrated solutions at least, cupric salt is less efficient than ferric salt. Comparing the details of the curves, however, it appears that the reaction with ferric salt is probably near a state of equilibrium; but the experiment with cupric salt suggests that, given a longer time, considerably more gold may be dissolved. It cannot be concluded, therefore, that the solvent action of a cupric salt would be less than that of a ferric salt, if a very much longer time were allowed to lapse before the loss of gold in the two experiments was ascertained, although the experiments suggest that this is probable.

The curves of Fig. 3 show that a dilution of the concentrated solution of cupric chloride and hydrochloric acid greatly decreases the amount of gold dissolved under the same conditions. For example, the solution containing 2 g. of copper as cupric chloride and 50 cc. of HCl (sp. gr. 1.178) dissolved 2.84 mg. of gold in 163 hr. Under the same conditions a solution of the same salts, but of one-half the concentration, dissolved only 0.23 mg. of gold in 163 hr. It thus appears that a dilution of the solution to one-half decreases its solvent action (2.84 divided by 0.23) to about one-twelfth. If the solvent were diluted to approximately the strength of mine-waters, it should be expected that the efficiency of cupric salt in these reactions would be almost immeasurably decreased. Indeed, the lower curve, *A*, in Fig. 3, strongly suggests this, and indicates also that the reaction with cupric salt at this concentration is nearing completion; for about half as much gold (0.11 mg.) was dissolved by this solution in 66 hr. as was dissolved in 163 hr. (0.23 mg.). It is improbable that the character of this curve would greatly change if the reaction continued over a period twice as long, and, projecting the curve to 14 days, in order that the solvent action of cupric salt may be compared with that of manganitic salt, it appears that in 14 days 0.48 mg. of gold would be dissolved in waters of this concentration,

assuming that the gold dissolved is in proportion to the time exposed, thus giving the advantage to cupric salt. The experiments with MnO_2 were carried on at about 18° C., and those with cupric salt at from 38° to 45° C.

The gold dissolved (experiment 16*b*) in the dilute solution with manganese was more than 10 times as much as that dissolved with the cupric salt (experiment 8). The hydrochloric acid (sp. gr. 1.178) with cupric chloride contained 34.99 per cent. of HCl and 34 per cent. of Cl. Disregarding the Cl introduced by cupric chloride, the solution used (25 cc. diluted to 125 cc.) contained 6.8 per cent. Cl. The solution with manganese dioxide (one-twenty-fifth normal) contained but 0.14 per cent. of Cl. The chlorine (in acid) in the experiment with copper was thus 49 times as much as the total chlorine in the experiment with manganese.

The amount of solution used in experiment 8, with cupric salt, was 2.5 times as much as the amount of solution used in experiment 16*b*, with manganese, but the area of gold exposed was not so great. In experiment 8 the gold was rolled to a thickness of 0.009 in. and cut into 1-mm. squares, whereas that used in experiment 16*b* was rolled to a thickness of 0.002 in., exposing areas of about 350 sq. mm. In the experiment with copper 250 mg. of gold was introduced, whereas only 150 mg. was introduced in experiment 16*b*. Correcting for areas exposed, a cupric solution 50 times as concentrated as the manganitic solution with respect to chlorine will dissolve about one-fifth as much gold where equal areas are exposed. In other words, the action with manganese appears to be more than 250 times as efficient as with cupric salt, even if it is assumed that further dilution would not decrease the solvent action of cupric salt in a geometrical ratio, as is indicated by the curves in Fig. 3. Comparing the end-points of curves *A* and *D*, Fig. 3, it is seen that a dilution of the solution to one-half decreases the solvent action with cupric salt to about one-twelfth. If further dilution to one-twenty-fifth normal HCl decreases the solvent action with cupric salt in this ratio, then the efficiency of the solution with cupric salt would be about $\frac{1}{1\,000\,000}$ as great as with MnO_2. It is thus shown that the efficiency of cupric salt compared with that of manganitic salt under these conditions is somewhere between 0.004 and 0.000001.

6. *The Amount of Chlorine Necessary for the Solution of Gold in the Presence of Manganese Compounds.*

In experiment 15 (*a*), with MnO_2, 0.01640 g. of gold was dissolved in 34 days with solution one-tenth normal with respect to chlorine. A solution with but 40 per cent. as much Cl (experiment 16*b*) dissolved 31 per cent. as much gold in 14 days as was dissolved in the more concentrated solution in 34 days. These results show that in 15 (*a*) conditions are probably approaching equilibrium and also that the solvent power of chlorine is approximately proportional to the amount present. That a weighable quantity of gold is dissolved when only a trace of chlorine is present is shown by experiment 13 (*b*), in which chlorine was introduced without intention.

7. *The Precipitation of Gold.*

Although gold is readily precipitated by organic matter, this reaction is not of great importance in igneous rocks. There ferrous sulphate is the chief precipitating-agent. Ferrous sulphate is formed by the oxidation of pyrite, but in the presence of oxygen and H_2SO_4 it becomes ferric sulphate, which does not precipitate gold. Below the water-table, where pyrite is more abundant and free oxygen less abundant, ferrous sulphate may persist in the mine-waters. Ferrous sulphate is so effective as a precipitant of gold that it is used for that purpose in metallurgical processes. Experiment 19, by W. J. McCaughey, shows that a minute amount of ferrous sulphate greatly decreases the solubility of gold, although it does not precipitate it completely. With excess of ferrous salt practically all of the gold is precipitated Don[40] has shown that many of the sulphate mine-waters of New Zealand and Australia contain abundant ferrous iron; and that such waters will first precipitate gold, but after oxidation will dissolve it.

Ferrous sulphate is formed in the upper part of a lode above the water-table; but owing to the open condition of that part of the lode, air is freely admitted and ferric sulphate forms, at the expense of ferrous sulphate and sulphuric acid. This reaction takes place almost instantaneously if MnO_2 is present (experiment 20), for ferrous sulphate and manganese dioxide are under

[40] *Trans.*, xxvii., 599 (1897).

these conditions incompatible. Manganese dioxide then not only releases the solvent for gold, but eliminates the salt which precipitates it. It is doubtful whether appreciable amounts of gold are ever carried far below the water-table in mines where the waters carry ferrous sulphate, but, in the presence of MnO_2, ferrous sulphate may be eliminated below the water-table.

When manganese dioxide takes part in the reactions by which, under the conditions named, gold is dissolved, transported and precipitated, the manganese salt is itself changed. At the surface pyrolusite, MnO_2, forms, for there an excess of oxygen prevails; and this mineral is commonly found in the gossan of manganiferous lodes. When solutions containing H_2SO_4 and NaCl react on MnO_2 there is a tendency to form $MnSO_4$, and some manganese goes into solution as sulphate, but salts of manganese with higher valence may also form. In this connection Dr. R. C. Wells has offered the following statement:

"In an acid solution containing some free chlorine, such as has been assumed to be effective in dissolving gold, there would also be a tendency towards the formation of permanganic acid. On the other hand, the production of the chlorine necessarily results in the reduction of the manganese compound. Now a manganous salt is known to react with permanganate to reproduce MnO_2 and this illustrates the tendency of manganese to pass with ease from one stage of oxidation to another. The precipitation of manganese will occur more and more as the solution loses its acidity. It is well established that manganous salts in an acid environment are very stable; but in neutral or alkaline solutions they oxidize more vigorously, one stage of their oxidation being the manganic salt which hydrolyzes into $Mn_2O_3.H_2O$ (manganite), with even greater ease than ferric salts into limonite.

"In these ways the migration of an acidic solution would result in the transportation of both gold and manganese. But in a region of basic, alkaline and reducing environment the manganese would be re-precipitated, the free acid neutralized, the chlorine absorbed by the bases and removed, and owing to the accumulation of the ferrous or other reducing salts, the gold would be re-precipitated."

V. The Transfer of Gold in Cold Solutions.

1. *Restatement of the Processes as Related to Secondary Enrichment.*

Every theory of secondary enrichment of the metals consists essentially of three parts: (*a*) solution, (*b*) transportation, (*c*) precipitation.

(*a*) As already stated, there is in the upper part of the ore-deposit, where oxidation prevails, abundance of ferric sulphate

and sulphuric acid. A little salt, NaCl, or other chloride, is generally present. The H_2SO_4, reacting upon NaCl, gives HCl, which, in the presence of MnO_2, gives nascent chlorine, which dissolves gold. Some manganese goes into solution as sulphate, but certain higher manganates are probably formed as well.

(*b*) This chemical system will move downward under hydrostatic head. If it comes into a zone containing pyrite it will react upon the pyrite, and in the oxidation of the latter more iron sulphates and acid will be formed. If manganese dioxide is present, or if permanganic acid had been formed, no gold will be precipitated, and the system, with gold still in solution, will move to greater depths before ferrous sulphate can become effective.

(*c*) But as the system moves downward, where no new sources of oxygen are available, the excess of acid is removed. There are many ways by which acidity is reduced along with these reactions, but the principal one is probably the kaolinization of sericite and feldspar. In these reactions sodium, potassium, calcium, magnesium, and other sulphates are formed from acid and silicates; the silica remaining as SiO_2 and kaolin; the alkalies and alkalic earth sulphates going into solution. As the acidity decreases, iron and manganese compounds tend to hydrolyze and deposit oxides. At this stage of oxidation $FeSO_4$ becomes increasingly prominent, and not only completely inhibits further solution of gold but becomes increasingly effective as a precipitant. Thus manganite is probably precipitated with gold. The fractures in the primary pyritic gold-ore below the water-level thus become coated with a manganiferous gold-ore, which may be very rich. The excess of oxygen which the system has carried down is used up in the manner indicated, and in this process limonite is formed, consequently the manganiferous gold-ore deposited in the fissures and cracks contains iron and kaolin as well as manganese oxides.

2. *Association of Gold with Manganese Oxides.*

Oxidized manganiferous ore frequently carries silver[41] without gold. In the oxidized zone such ore should be common;

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[41] *Mining and Scientific Press*, vol. xciv., No. 25, p. 796 (June 22, 1907).

but in the sulphide zone different relations, according to the requirement of the theory, should generally be shown. In this zone the manganese acts not so much as a solvent for gold, but rather as an agent which delays precipitation by converting the ferrous sulphate, which precipitates gold, to ferric sulphate. The gold has presumably been dissolved higher up, but it traveled downward in solution in cracks in the primary sulphide ore. It would be expected that the deeper-seated manganiferous ore, unlike the lean ore in the oxidized zone, would be rich in gold. S. F. Emmons informs me that there is a common feeling among the miners in Colorado that manganese is a very good sign of rich ore. The same feeling exists in the minds of many prospectors elsewhere. F. L. Ransome says[42] that in the Camp Bird and Tomboy mines black oxide of manganese occurs in the deeper workings (year 1901), and usually indicates good ore. In these cases, according to Ransome, "The oxide appears to be associated with post-mineral fracturing . . . and to have been deposited later than the bulk of the ore." In general, the gold-deposits near Telluride and Ouray show very little secondary enrichment and the primary ore is rich enough to pay handsomely, but the small rich manganese streaks may be rationally explained by the processes indicated. In the deposition of chalcocite ferrous sulphate is formed, and this would readily precipitate the gold if any were held in the solution. The relation of chalcocitization and deep-seated precipitation of gold is discussed on p. 42.

In the oxidized zone small bunches of very rich manganiferous gold-ore are often found. I have seen such ore above the sulphide zone in certain camps in Nevada. Such bunches of rich ore were probably formed when they were surrounded by sulphides, but were overtaken by the oxidized zone, which moves progressively downward, and the gold in the rich ore has not yet been dissolved. Such ores should in general be more abundant and richer in the lower part of the oxidized zone than near the apex, where they have been exposed for longer periods to the solutions dissolving gold. They may be compared with the rich partly-oxidized chalcocite which appears near the surface in certain copper-mines. Such ore

[42] A Report on the Economic Geology of the Silverton Quadrangle, Colorado, *Bulletin No.* 182, *U. S. Geological Survey,* p. 101 (1901).

remains above the water-level because the table has been depressed more rapidly than the copper sulphide has been dissolved. The mutual relation of these processes is discussed by W. Lindgren in his monograph on the copper-deposits of the Clifton-Morenci district, Arizona.[43]

3. *The Oscillating, Descending, Undulatory Water-Table.*

The terms "water-table" and "level of ground-water" are generally used to describe the upper limit of the zone in which the openings in rocks are filled with water. This upper limit of the zone of saturation is not a plane, but a warped surface. It follows in general the topography of the country, but is less accentuated. It is not so deep below a valley as below a hill, but it rises with the country towards the hill-top and in general is higher there than in the valley. Nor is it stationary. In dry years it is deeper than in wet years, and in dry seasons it is deeper than in wet seasons. The difference of elevation between the top of this zone in a wet year and in a dry year is normally greater under the hill-top than on the slopes and in the valleys. In mines where the ground is open the level of ground-water probably changes with every considerable rain. Consequently, there is a zone above ground-water in dry periods but below it in wet periods, and in hilly countries this may be of considerable vertical extent. Thus the water-table oscillates, though in general moving downward with degradation of the land-surface. It is in this zone of oscillation of the water-table that chemical activity is most varied. Without any change in the character of the drainage or of the more-constant conditions controlling the water-circulation, the chemical composition of the solutions affecting this zone may change from season to season. They may at one time be ferric sulphate or oxidizing waters and at another time ferrous sulphate or reducing waters, since, after a wet season, the ferrous sulphate waters from below would tend to rise, after dilution with fresh water added by the rains. Consequently, the minerals of this zone may include, besides the residual primary and secondary sulphides, the oxides, native metals, chlorides, sulphides, carbonates, etc. Between the top of this zone and the surface or the apex of the deposit chemical activity is probably slow, be-

[43] *Professional Paper No.* 43, *U. S. Geological Survey*, p. 232 (1905).

cause there is a scarcity of sulphides and other easily-altered minerals to supply the salts upon which the chemical activity of ground-water in a large measure depends. As the country is eroded, this zone also descends; and if a mineral or metal persists long enough, the upper limit of the zone of active change passes below it. The mineral is thus "marooned," and, not being exposed to mineral-laden waters, it may ultimately be exposed at the outcrop of the deposit.

4. *The Several Successive Zones in Depth.*

As is clearly set forth by S. F. Emmons, W. H. Weed and others, many metalliferous lodes, when followed from the surface down the dip, show characteristic changes. Below the outcrop, the upper part of the oxidized portion of the lode may be poor. Below this there may be rich oxidized ores; still farther down, rich sulphide ores; and below the rich sulphides, ore of relatively low grade. Such ore is commonly assumed to be the primary ore, from which the various kinds of ore above have been derived. The several types of ore have a rude zonal arrangement, the so-called "zones" being, like the water-table, highly undulatory. They are related broadly to the present surface and to the hydrostatic level, but are often much more irregular than either; for they depend in large measure on the local fracturing in the lode which controls the circulation of underground waters. Any zone may be thick at one place and thin, or even absent, at another. If these zones are platted on a longitudinal vertical projection, it is seen that the primary sulphide ore may project upward far into the zone of secondary sulphides, or into the zone of enriched oxides, or into the zone of leached oxides, or may even be exposed at the surface. The zone of secondary sulphide enrichment (which is not everywhere present) may project upward far into the zone of rich oxidized ore, or into the zone of leached oxides, or may outcrop at the surface. The zone of sulphide enrichment nearly always contains considerable primary ore, and very often the secondary ore is merely the primary ore containing in its fractures small seams of rich minerals. The zone of enriched oxides is generally found above the water-table when the latter is at the lowest. This zone often extends to the outcrop. Indeed, it is at such places that most mines are discovered, for in districts not

known to contain metalliferous deposits a lean or barren outcrop is generally not extensively explored by prospectors. In regions of rapid erosion, and especially of rugged topography, the conditions for the exposure of rich oxides, or even rich sulphides or primary ore, are more favorable. In places along the outcrop of a deposit where erosion is rapid the richer oxidized or sulphide ores may be exposed, whereas in other places, protected from erosion, and therefore exposed longer to solution, the same outcrop is frequently leached. It is evident that the amount of metal remaining in the upper part of the oxidized zone and at the outcrop depends upon the ratio between the rate at which the metal is dissolved, and the rate at which the valueless constituents are dissolved and removed. Under certain conditions gold is removed very slowly, and the removal of valueless constituents may effect a concentration at the very apex of the lode; while under other conditions, favorable to the solution of gold, it is removed more rapidly than the valueless constituents (such as silica and iron) and, in consequence, the apex and the upper portion of the zone below it are leached. In a country not subject to erosion it would be supposed that the outcrops of manganiferous lodes would be everywhere leached; but rapid erosion may remove the upper part of the lode before it is completely leached, and, under favorable conditions, placers accumulate from the débris of the apex.

It thus appears that all of these zones except that of the primary ore are, broadly considered, continually descending; so that ore taken from the outcrop may represent what was once primary ore; afterwards, enriched sulphide ore; still later, oxidized enriched sulphide ore; later still, leached oxidized enriched sulphide ore; and finally become the surface-ore. Through more rapid erosion at some particular part of the lode, any one of these zones may be exposed; and hence an outcrop-ore of any character is possible. Consequently, longitudinal assay-plans, showing the changes of value in depth, though highly suggestive, and especially so when gold and silver are shown separately, are supplemented by studies of the paragenesis and by physiographic studies, in order that the approximate rate of erosion of the lode at various places may be known. In the absence of such knowledge, it is generally im-

possible to tell the genesis of a particular sample of ore from a mine, although this may sometimes be done. When all the data are assembled, however, greater confidence may be placed in the conclusion, since all the factors in the problem are intimately related.

5. *Criteria for the Recognition of Secondary Enrichment.*

I shall not attempt to review all the criteria for the recognition of secondary enrichment. They involve practically all available data relating to the geology and physiography of the region, as well as the observed characteristics of its ore-deposits. But each group of deposits may be studied with certain general criteria in view. Among these are: (1) the vertical distribution of the richer portions of the lode with respect to the present surface and to the level of ground-water; (2) the mineralogy of the richer and poorer portions of the deposit, and the character and vertical distribution of the component minerals; (3) the paragenesis, or the structural relations shown by the earlier ore and that which has been introduced subsequently.

In applying these principles, it should be remembered that circulation is generally controlled by post-mineral fracturing; that the changes depend upon climate and rapidity of erosion, and are affected by regional changes of climate, etc. Although the mineralogy of the ore is a useful aid, there are many minerals which are precipitated from cold solutions and also from ascending hot solutions, and there are many others, the genesis of which is uncertain. Of the minerals formed in the zone of secondary sulphide enrichment, few, if any, are known positively to form under such conditions only. There are some, however, such as chalcocite and covellite, which nearly everywhere are clearly of secondary origin. Ruby-silver is frequently, but not always, secondary. Other minerals, such as chalcopyrite, bornite, argentite, etc., have no definite indicative value unless their occurrence suggests that they are later than the primary ore. Where minerals, known to have formed elsewhere by processes of secondary sulphide-enrichment, are clearly later than primary ore, there is a strong presumption that they were deposited by cold descending waters. If it can be shown, in addition, that they do not extend to the bottom of the mine, but are related to the present topography of the country, then

this presumption may be regarded with considerable confidence as confirmed.

Where paragenetic evidence suggests secondary enrichment, it should be determined whether the later minerals are those commonly formed by secondary processes, for, as shown by Weed and others, certain minerals, such as enargite and rhodochrosite, may be deposited by ascending solutions in openings in older ore-bodies.

With respect to gold, the problem is difficult, because the native metal is the only stable gold-mineral known to be deposited from cold dilute solutions. Consequently, the applicable criteria are limited; and the vertical distribution of the richer ore, though suggestive, is not in itself conclusive. Lindgren and Ransome, in their studies at Cripple Creek, have shown that the richer ore-bodies may have in general a relationship to elevation, where there is little or no evidence of deep-seated secondary enrichment. The maximum deposition by ascending hot waters may be greater at one horizon than at another; and the rich ore, though showing broadly certain variations with depth, is in no way related to the water-table. If, however, it can be shown that rich seams of ore cross the primary ore and do not extend downward as far as the bottom of the primary ore, but are related to the present topography of the country, and if it is known that the associated minerals which fill such openings are those which may be deposited by cold waters, the evidence of their secondary origin is practically conclusive. As already shown, seams of gold with limonite and manganese oxides occur in such relations. Similar ore frequently contains chalcocite and argentite also. Such occurrences could with great confidence be attributed to descending waters; and since it is known that they are commonly related to the present surface, a fair presumption is that they will disappear in depth.

In the practical application of such reasoning to gold-bearing deposits it will sometimes be necessary to discriminate between the oxidized manganiferous gold-ore which has resulted simply from the oxidation of a primary manganiferous ore like one containing rhodochrosite, and that which has been deposited in fractures in the sulphides lower down. In other words, it is desirable to know whether rich manganiferous ore in the upper

part of a mine is residual from a primary ore-body, and therefore will probably prove extensive, or represents the result of concentration under more deeply seated conditions after the manner indicated above. This discrimination may be easy in the sulphide zone, where the fractures with rich manganiferous ore are clearly shown; but in the oxidized zone one must rely upon the shape and distribution of the rich bunches. If they are related to cracks in the mass of the oxidized ore, the inference is warranted, in the absence of other evidence, that they are residual secondary ore, and, being genetically related to the present topographic surface, are limited.

The tellurides and selenides of gold are seldom or never deposited from cold solutions; hence native gold is, as already stated, the only gold-mineral which may be so deposited. But native gold is deposited by primary processes also, and is by far the most abundant gold-mineral so deposited. Consequently, in distinguishing between primary gold and gold deposited by cold solutions, one must rely upon associated minerals. When secondary chalcocite or certain secondary silver-minerals are deposited, the attendant reactions precipitate gold. Consequently, the richer bunches of gold-ore in the oxidized zone, residual from secondary ore formed under the deeper-seated conditions, may carry also considerably more copper and silver than the primary ore. But copper, and (unless cerargyrite is formed) silver also, are more readily leached than gold, even when manganese is present. Hence, the evidence of this character may have been destroyed.

With respect to other minerals associated with the secondary gold-ore, we are not warranted, in the present state of our knowledge, in drawing definite conclusions. From the nature of the reactions, I think it may be possible to show that manganite, $Mn_2O_3.H_2O$, is, under conditions of incomplete oxidation, more often associated with the rich gold in such relations than pyrolusite, MnO_2,; for, as already observed, the lower oxide is more likely to be precipitated than the higher, when secondary gold is deposited under deep-seated conditions. But under oxidizing influences the manganese oxides change their character so readily that this criterion, if it has any value, is probably not applicable to ores in the upper part of the oxidized zone, where they have been exposed to more highly oxygenated

waters for a longer time. I make these suggestions with respect to the character of the manganese oxides associated with the rich ore, not because I think the reactions which precipitate manganese are well enough understood to give a positive paragenetic value to the oxidized manganese-minerals themselves, but in the hope that others will ascertain and report the character of the manganese oxide associated with gold in the deeper zone and in the residual products from that zone. The streak of manganite is reddish brown, sometimes nearly black, whereas the streak of pyrolusite is black or bluish black; but mixtures and pseudomorphs of the minerals occur, and it is sometimes almost impossible to determine which oxide is present.

In some gold-veins[44] the vein-cavities near and even a considerable distance below the oxidized zone are filled with a brown or black mud, which is frequently very rich. It is not safe to assume that the gold in such cavities was carried to its present position in solution and precipitated by ferrous sulphate. The fine pulverulent ore which collects in the cracks is rich in gold, and may have been carried downward in suspension. But such ore will generally show a horizontal stratification, which will seldom be shown by the ore deposited from solution. As suggested above, manganite, rather than pyrolusite, is probably formed when gold is precipitated. Such mud, deposited from suspension, may contain either pyrolusite or manganite or both; but it is rational to assume that the mud formed by precipitation in the deeper zone carries very little pyrolusite, but is mainly manganite.

6. *Lateral Migration of Manganese-Salts from the Country-Rock to the Ore.*

Clarke's analyses[45] show that igneous rocks carry an average of 0.1 per cent. of manganese oxide, and many basic rocks carry from 0.2 to 0.9 per cent. Where basic dikes have cut an ore-body, they doubtless contribute manganese to the waters circulating in the deposit. The ore of the Haile mine, in South Carolina, is cut by basic rocks; and the ore-bodies of the Delamar mine, in Nevada, are crossed by a basic dike. Both of these de-

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[44] Cripple Creek, *Professional Paper No.* 54, *U. S. Geological Survey*, p. 199 (1906).

[45] *Bulletin No.* 330, *U. S. Geological Survey* (1908).

posits show secondary enrichment of gold; and in both the better ore is found along the dikes. In general, however, the manganese from the country-rock cannot safely be assumed to have migrated extensively into the ore-deposit, for many analyses of mine-waters do not show manganese; but where manganiferous rocks are intimately fractured and filled with seams of ore it would be supposed that the reactions requiring manganese could take place.

The experiments of Dr. Eugene C. Sullivan, performed at the request of S. F. Emmons, in the investigation of another problem, have an important bearing here; and Mr. Emmons has kindly permitted me to publish them in advance of his own paper. In so doing, I have abridged somewhat the statements of Dr. Sullivan.

A sample of the lower white porphyry from the Thespian mine, Leadville, Colo., was finely ground and treated with carbonic acid and with sulphuric acid ; the rock contained 0.8 per cent. of iron and 0.033 per cent. of manganese. The ratio is about 24 to 1.

Carbonic Acid.—20 g. of the porphyry was taken in 40 cc. of water, and carbon dioxide was passed into the mixture for some hours. In 20 cc. of the solution 0.03 mg. of manganese were found and no iron. The results are probably correct for manganese to 0.01 mg. Less than 0.01 mg. of iron would have been detected, if present To preclude the possibility that the solution of manganese was facilitated by its reduction with metallic iron introduced from the hammer in pounding up the sample, another portion was similarly treated after metallic iron and magnetite had been removed by a hand-magnet. In this case 0.1 mg. of manganese and 0.02 mg. of iron were found in 20 cc. of solution.

Sulphuric Acid.—20 g. of the powdered porphyry stood over night in contact with 40 cc. of one-tenth normal sulphuric acid (0.196 g of H_2SO_4 in 40 cc). This has roughly the same molecular concentration as a saturated solution of carbon dioxide. The filtrate, 20 cc., contained 1.05 mg. of iron, all in the ferrous condition, and 1 mg. of manganese The experiment was repeated under the same conditions, except that contact between the rock-powder and the acid was of but a few minutes' duration ; 1.20 mg. of iron, practically all ferrous, and 0.90 mg. of manganese were found in 20 cc. of solution. One-tenth of a milligram is about the limit of accuracy in these cases.

Potassium Sulphate.—Neither iron nor manganese could be detected in the solution after treatment of the rock-powder with potassium sulphate.

Manganese is therefore more readily extracted from the rock than iron under surface-conditions ; for, although it is present in the ratio of only 1 : 24 as compared with iron, yet carbonic acid takes out more than three times as much manganese as iron, and sulphuric acid gives a ratio of about 1 : 1.[46]

As to the precipitation of the two metals from a mixture of their salts in solu-

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[46] Penrose, The Chemical Relation of Iron and Manganese in Sedimentary Rocks, *Journal of Geology*, vol. i., pp. 356 to 370 (May–June, 1893). Vogt, Bog Manganese-Ores, *Zeitschrift für practische Geologie*, vol. xiv., p. 217 (July, 1906).

tion, the following experiment shows that ferrous compounds are more readily oxidized and precipitated than manganous compounds. Ferrous sulphate solution and manganous sulphate solution were mixed in equi-molecular quantities (50 cc. containing 2 mg-molecules of each, *i e*, 0.112 g. of iron and 0 110 g of manganese), with sufficient powdered calcite (Iceland spar) to react with one of the metals (0 200 g.-molecule of calcite). During four weeks the mixture, in a roomy flask, was occasionally shaken, the stopper at the same time being removed for a moment to allow free access of air. At the end of that time all but 1 5 mg of the iron had been precipitated, while the manganese was in solution in practically the same quantity as originally Calcite, however, when in contact with manganous salts alone, in the presence of air, will precipitate the manganese as a higher oxide or hydroxide, especially at elevated temperature.

It thus appears that some manganese is probably contributed to the ore-deposits from the country-rock. I believe, however, that such additions are small, except where space-relations of ore and country-rock are peculiarly favorable. In the upper parts of a vein the circulation is in general downward, and is controlled very closely by fractures, which are more abundant in the upper zone, where the rocks are in general more extensively shattered. Gouge-seams on the walls would also limit the circulation, and tend to keep the vein free from waters of the country-rock. Where calcite or other carbonates are present to precipitate the small amount of manganese in the solutions, one would suppose that the opportunities for slight additions would be increased. Manganese carbonate is less soluble than calcite and the latter could, under favorable conditions, be replaced by manganese compounds. One part of calcium carbonate is soluble in 1,428 parts of water saturated with carbon dioxide, while one part of manganese carbonate is soluble in 2,000 parts of water so saturated.[47]

In my own experience I have found only trivial stains of manganese in those lodes where it was not present in the gangue of the primary ore; and, in view of its wide distribution in igneous rocks, I believe that the lateral migration of manganese into the ore under the conditions which generally prevail is very subordinate. Though the amount so contributed may facilitate the solution of gold, it is probably inadequate to form sufficient higher manganates or similar salts to suppress effectively the action of ferrous sulphate. Under such conditions the gold could not travel to the reducing-zone below the water-level, but would be precipitated practically at the place where it had been dissolved.

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[47] *Lassaigne, in Comey's Dictionary of Solubilities* (1896).

7. *Concentration in the Oxidized Zone.*

The concentration of gold in the oxidized zone near the surface, where the waters remove the valueless elements more rapidly than gold, is fully treated by T. A. Rickard in his paper on the Bonanzas in Gold-Veins.[48] Undoubtedly this is an important process in lodes which do not contain manganese, or in manganiferous lodes in areas where the waters do not contain appreciable chlorine. In the oxidized zone it is sometimes difficult to distinguish the ore which has been enriched by this process from ore which has been enriched lower down by the solution and precipitation of gold, and which, as a result of erosion, is now nearer the surface. It cannot be denied that fine gold migrates downward in suspension; but in all probability this process does not operate to an important extent in the deeper part of the oxidized zone. If the enrichment in gold is due simply to the removal of other constituents, it is important to consider the volume- and mass-relations before and after enrichment, and to compare them with the present values. In some cases, it can be shown that the enriched ore occupies in the lode about the same space as was occupied before oxidation. Let it be supposed that a pyritic gold-ore has been altered to a limonite gold-ore, and that gold has neither been removed nor added. Limonite (sp. gr. from 3.6 to 4), if it is pseudomorphic after pyrite (sp. gr. from 4.95 to 5.10) and if not more cellular, weighs about 75 per cent. as much as the pyrite. In those specimens which I have broken, cellular spaces occupy in general about 10 per cent. of the volume of the pseudomorph. With no gold added, the ore should not be more than twice as rich as the primary ore, even if a large factor is introduced to allow for SiO_2 removed and for such cellular spaces.

Rich bunches of ore are much more common in the oxidized zone than in the primary sulphides of such lodes. They are present in some lodes which carry little or no manganese in the gangue, and which below the water-level show no deposition of gold by descending solutions. Some of them are doubtless residual pockets of rich ore which were richer than the main ore-body when deposited as sulphides, but others are very

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[48] *Trans.*, xxxi, 198 to 220 (1901).

probably ores to which gold has been added in the process of oxidation near the water-table by the solution and precipitation of gold in the presence of the small amount of manganese contributed by the country-rock. In view of the relations shown by the chemical experiments it is probable that very little manganese will accomplish the solution of gold, but, as already stated, it requires considerably more manganese to form appreciable amounts of the higher manganese-compounds which delay the deposition of gold, suppressing its precipitation by ferrous sulphate. In the absence of larger amounts of the higher manganese-compounds, the gold would probably be precipitated almost as soon as the solutions encountered the zone where any considerable amount of pyrite was exposed in the partly-oxidized ore; for Buehler and Gottschalk have lately shown that oxygenated solutions attack pyrite and dissolve it in a comparatively short time, and McCaughey has shown that even traces of the ferrous sulphate thus formed precipitate gold almost immediately. From this it follows that deposits showing only traces of manganese, presumably supplied from the country-rock, are not enriched far below the zone of oxidation.

8. *Vertical Relation of Deep-Seated Enrichment of Gold to Chalcocitization.*

In several of the great copper-districts of the West (see group 3, p. 50) gold is a by-product of considerable value. In another group of deposits, mainly of Tertiary age (see group 4, p. 51), and younger than the copper-deposits, silver and gold are the principal metals, and copper, when present, is only a by-product. But in some of these precious-metal ores chalcocite is, nevertheless, the most abundant metallic mineral, often constituting 2 or 3 per cent. of the vein-matter. Frequently it forms a coating over pyrite or other minerals. Some of this ore, appearing in general not far below the water-table, is fractured, spongy quartz, coated with pulverulent chalcocite. It frequently contains good values in silver, and more gold than the oxidized ore or the deeper-seated sulphide ore. Clearly, the conditions which favor chalcocitization are favorable also to the precipitation of silver and gold.

The exact chemical reaction which yields chalcocite is not

known. At 100° C., according to Dr. H. N. Stokes,[49] the reaction with pyrite is probably about as follows:

$$5\,FeS_2 + 14\,CuSO_4 + 12\,H_2O = 7\,Cu_2S + 5\,FeSO_4 + 12\,H_2SO_4.$$

In the cold, the reaction may differ in details, but without doubt much ferrous and acid sulphate is set free. Attendant reactions confirm this statement; for, if calcite is present, gypsum is formed by the reaction of H_2SO_4 on lime carbonate; and, if the wall-rocks are sericitic, kaolin is formed by the acid reacting upon potassium-aluminum silicate, the potash going into solution as sulphate. The abundant ferrous sulphate must quickly drive the gold from solution, and it apparently follows that there may be no appreciable enrichment of gold below the zone where chalcocitization is the prevailing process. In deposits such as those of disseminated chalcocite in porphyry, where the chalcocite occurs in flat-lying zones related to the present surface, and where the ore from which chalcocite was derived carried gold, and suitable solvents were provided, there should be a comparatively even distribution of gold, which should increase and decrease with the chalcocite of the secondary ore. A different ratio of values should be found in the oxidized low-grade capping above the chalcocite, for the solution of gold, even under the most favorable conditions, appears to lag behind the solution of copper, and this should be more marked in these deposits, since in all available analyses the porphyries are low in manganese, and rhodochrosite is not noted in the primary ore. I am informed that a fairly-constant ratio between copper and gold is very noticeable in the disseminated deposits at Ely and at Bingham. That whatever gold is present in the rock below chalcocitized pyrite is not a result of deposition from cold solution, is reasonably certain under the conditions named.

9. *Vertical Relations of Silver-Gold and Gold-Silver Ore in Deposits Carrying Both Metals.*

This paper will not discuss in detail the processes of secondary enrichment of silver-deposits—a subject already treated in

[49] Unpublished MSS. quoted by Lindgren in *Professional Paper No.* 43, *U. S. Geological Survey*, p. 183 (1905), and in Weed's translation of Beck's text-book.

our *Transactions* by S. F. Emmons, W. H. Weed, and C. R. Van Hise. There are, however, certain deposits mainly associated with Tertiary rocks, in which both silver- and gold-values are important. Examples are the Comstock lode, Tonopah, Tuscarora, etc. Where physical conditions are favorable, deposits of this type should show in general a concentration of gold at certain horizons, and of silver at other horizons, depending upon the composition of the mine-waters and other factors. The determination, in such mines, of the principles controlling the mutual relations, especially in the deeper zones of the gold-silver and the silver-gold ore-bodies, would have great practical value. So far as I know, no record of experiments with solutions containing both sulphates and chlorides and a mixture of gold and silver is available. The solubilities of silver-salts lately determined by Kohlrausch (quoted by Alexander Smith) are suggestive. He found that at 18° C. a saturated aqueous solution of $AgSO_4$ contains 5.5 g. per liter; but at this temperature water holds in solution only 0.0016 g. of silver chloride per liter. That silver is held in solution by mine-waters carrying sulphates and chlorides was shown by J. A. Reid.[50] Such waters in a Comstock mine carried about 188 mg. of silver and 4.15 mg. of gold in a ton of solution.

The effect, on a manganiferous silver-ore, of a solution carrying chlorides would be to liberate chlorine, which would react with silver to form "horn-silver." This would be fixed in the manganiferous ore, and such a silver-ore would be comparatively stable. The oxidized manganiferous silver-ore at Leadville, Colo.,[51] and at Neihart, Mont., in which silver is generally supposed to be carried largely as chloride, may have originated in this manner. On the other hand, rich ores could hardly be formed where the solutions carried abundant sulphuric acid and little or no chlorine, for the soluble silver sulphate would be formed, and the manganiferous ore leached. To determine the genesis of such manganiferous ore, it is desirable to know the silver-content of the primary rhodochrosite, for, as indicated above, two interpretations of the phenomenon are otherwise possible.

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[50] *Bulletin of the Department of Geology, University of California*, vol. iv., No. 10, p. 193 (1904–06).

[51] S. F. Emmons, *Monograph No. XII., U. S. Geological Survey*, p. 562 (1886).

As pointed out by Penrose,[52] silver chlorides are formed extensively in arid countries, at or very near the surface. Frequently the workable ore gives out a few feet down. At many places in Nevada, the so-called "chloriders" stripped the surface over considerable areas; but where the unoxidized ore was encountered the mines were abandoned. At some of these places the chloride ore carried little, if any, more gold than the unoxidized, unprofitable silver-gold ore below. The primary ore of many of these deposits carries relatively little pyrite; and the inspection of a number of them gives the impression that the siliceous ores are more favorable than the more highly pyritic ores to the formation of a surface chloride zone. Manganese oxides are not necessary for the formation of the chloride zone. In many of them manganese is absent. If it is present in appreciable amount, if the physical conditions for a downward circulation in the lode are favorable, and if the primary ore carries gold, it would be reasonable to expect an enrichment of gold below the zone of the chloride-enrichment of silver. In the presence of strong acid sulphate waters, silver, like gold, is dissolved from the outcrop; and in some mines, where both metals are present in important quantities, the outcrop and the oxidized zone for a short distance below are leached of both silver and gold.

The migration of both metals with selective solution and precipitation is suggested by the relation of silver-gold and gold-silver ore-bodies on the Comstock lode. The Comstock lode, which has produced more than $200,000,000 silver and $150,000,000 gold, is a broad fault-zone in late Tertiary rocks. The ore-shoots occur here and there in this zone, which is developed more than 4,000 ft. below the surface.[53] Since the deposits were formed there has been extensive fracturing. In the lode there are great bodies of "sugar" quartz which are due, according to Becker,[54] to this movement. Over considerable spaces one cannot obtain fragments of rock as large as one's fist

[52] *Journal of Geology*, vol. ii., No. 3, p. 314 (Apr.–May, 1894).

[53] Clarence King, *Geological Exploration of the 40th Parallel*, vol. iii., *Mining Industry* (1870); John A. Church, *The Comstock Lode* (1879); G. F. Becker, *Monograph No. III., U. S. Geological Survey* (1882); J. A. Reid, *Bulletin of the Department of Geology, University of California*, vol. iv., No. 10, pp. 177 to 199 (1904–06).

[54] *Op. cit.*, p. 272.

which do not show fissures. There were clearly two periods of movement, one before the deposition of the primary ore and one following it. The latter movement, mainly parallel to the lode, gave conditions for an active circulation of water after the primary deposition. According to Dr. Becker, "it is possible that the seams of rich ore in the great bonanza represent a deposition posterior to the final cessation of movement," and "it is also by no means impossible that some of the richer ores have been redeposited, forming at the expense of surrounding bodies of lower grade."[55] As already remarked, analysis of the vadose water of the Comstock shows that it contains both gold and silver. It is noteworthy that this water contains much manganese, presumably as sulphate. Some associated placers were developed, but they are of very subordinate value compared with that of the lode. Oxidation extended downward as far as 500 ft. According to Clarence King,[56] "a zone of manganese oxide occupies the entire length of the lode from the outcrop 200 ft. down." The upper part of this manganiferous zone was probably not of high grade in general, especially in the uppermost portions. I infer that the outcrop and the ore immediately below were in general not so rich as the ore lower down. The longitudinal projections[57] show that many of the stopes carried from below stop some distance below the surface.

Von Richthofen (quoted by Becker) says that "the proportion of gold to silver decreased during the early period of working the lode, but is now (1865) on the increase again." Presumably, silver at the very surface was leached more rapidly than gold. The vadose waters, as shown by Reid,[58] are rich in ferric sulphate; and his analyses, as well as others, show the presence of chlorides in appreciable amounts. The conditions appear to have been favorable for the solution of both silver and gold in the upper levels, even in the comparatively short geological period which has elapsed since the primary ores were deposited. The bonanza ore below consisted largely of stephanite, polybasite, argentite, and other

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[55] *Monograph No. III., U. S. Geological Survey*, p. 273 (1882).

[56] *Geological Exploration of the* 40*th Parallel*, vol. iii., *Mining Industry*, p. 75 (1870).

[57] Becker, *op. cit.*

[58] *Bulletin of the Department of Geology, University of California*, vol. iv., No. 10, pp. 177 to 199 (1904–06.

dark, rich silver-minerals, and in places, according to Dr. Becker, appeared to fill fractures which involved the primary ore. It is well known that this rich ore was more abundant in the upper than in the lower levels. When I visited the district in 1907 I was informed by the foreman at the Consolidated Virginia that large bodies of the hydrothermally-altered porphyry on the Sutro level and below, which contained considerable pyrite, etc., carried also considerable gold and silver, although below the limit of profitable mining. Very few of the ore-bodies which had been worked at those levels were then accessible. The deposits in the upper levels yielded, according to Richthofen, from $70 to $107 a ton, whereas in later years the average value of the ore was not more than $37.

It thus appears that the evidence of the Comstock lode, from the surface down, is favorable to the hypothesis that extensive solution and deposition of gold and silver has taken place, while it is insufficient to show to what extent the great bonanzas owed their values to such processes. In ores formed so near the surface, there is always the possibility that ascending hot waters deposited the maximum portion of their gold and silver at the horizon where they encountered cold oxygenated solutions. Sulphate may form, in such mixtures, and ferrous sulphate tends to drive both gold and silver out of solution. The proportion of gold to silver was presumably higher in the upper part and in the lower part of the lode than in the middle portion. When Richthofen made his report, he estimated that the lode had produced, to the close of 1865, $15,250,000 of gold and $32,750,000 of silver (gold equals 47 per cent. of the silver); whereas Becker reports the amount recovered from 1865 to 1881 as $87,121,988 of gold and $105,548,157 of silver (gold equals 83 per cent. of the silver). If much of the change is due to reworking by descending waters, the greater gold-values in the upper portions of the bonanzas indicate that gold was dissolved less readily than silver, and silver precipitated less readily than gold, in the sulphate-rich water of this mine.

The relation of "horn-silver" to the surface is different from that shown in the "chloride mines" mentioned above. According to Clarence King,[59] silver chloride is accidental, although

[59] *Op cit.*, p. 82.

rare small crystals were found at the outcrop in the Gold Hill group. It occurred, however, at the 900-ft. level of the Yellow Jacket, where, judging from the descriptions, it was present in considerable amount.

Ferrous sulphate precipitates both silver and gold in acid solutions. The precipitation of gold is, however, many times more rapid and more effective than that of silver. Where silver chloride is not precipitated, one would suppose that cold solutions would transfer the silver to greater depths than gold. Since silver chlorides are not abundant (King), this, if no other, is an argument for the hypothesis that the lower-grade gold-silver ores ($37 a ton or less, Becker [60]), which were worked in the levels below the great silver-gold bonanzas, were in the main primary. These ore-bodies were not accessible to me when I visited the Comstock mines, and the speculation is not based on paragenetic evidence.

The relations of silver-gold to gold-silver ores in the Exposed Treasure mine, near Mojave, differ from those at the Comstock. In the surface-zone, horn-silver has been formed in considerable amount. The proportion of gold to silver in this zone is 1 : 72 (weight). In the lower friable siliceous ores, the proportion of gold to silver is 1 : 12; and in the sulphide ores below the water-table, where the gold-content had increased 150 per cent. above the average in the friable siliceous ores, the proportion of gold to silver was as 1 to 2.[61] The Exposed Treasure ores are, like those of the Comstock, manganiferous.

It thus appears that there are two types of enrichment in deposits of manganiferous gold- and silver-ores. In one of them silver chloride is concentrated in the manganiferous oxidized ores of the upper levels, and gold is concentrated below. In the other, silver chloride is subordinate, while both gold and silver are concentrated below the oxidized zone. Possibly the difference could be explained if the amount of chlorine were determined in the waters of deposits of both types. Silver chloride is soluble in an excess of alkaline chlorides. Those deposits in which horn-silver is not present may have been leached by waters unusually rich in chlorides.

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[60] *Monograph No. III., U. S. Geological Survey*, p. 18 *et seq.* (1882).

[61] Courtenay DeKalb, *Trans.*, xxxviii., 319 (1908).

VI. Review of Mining-Districts.

The purpose of this inquiry is to ascertain whether the ore-deposits of the United States give evidence that gold is more readily transferred in manganiferous deposits than in deposits which do not contain manganese, a hypothesis suggested by the chemistry of the processes of solution and precipitation.

1. If gold is more readily dissolved in manganiferous deposits, it would be supposed that placers form less readily from pyritic manganiferous lodes than from lodes containing no manganese. If, in areas where the waters carry appreciable chlorine, placers have formed as extensively from such lodes as from lodes free from manganese, then the hypothesis fails.

2. The manganiferous lodes, in areas of chloride waters, as in the undrained areas of the Great Basin, should in general show less gold at the outcrop and in the upper portion of the oxidized zone than below. In silver-gold deposits, however, silver, on account of the insolubility of the chloride, may remain, or be concentrated, in the oxidized manganiferous zone. Bunches of rich gold-ore carrying oxidized manganese in the oxidized zone are not necessarily fatal to the theory; for, as already stated, these are probably residual from the zone of secondary enrichment. An extensive enrichment in gold of the oxidized manganiferous ores at the surface, which are shown not to be residual from the zone of secondary ores, would indicate that the selective processes lack quantitative value, if the waters carry chlorine, and if the primary ores, from which the manganiferous oxidized ores are derived, carry appreciable pyrite to supply sulphate.

3. If in certain lodes gold migrates below the water-table, it should be precipitated quickly by ferrous sulphate. But MnO_2 converts ferrous sulphate to ferric sulphate, which does not precipitate gold. Hence, MnO_2 favors the solution of gold, and converting the ferrous salt to ferric sulphate removes the precipitant. Consequently, if auriferous lodes show enrichment in the deeper zone but related to the present surface of the country, the manganiferous lodes should, the other favorable conditions provided, show greater differences in values with respect to gold than lodes free from manganese.

Gold-Provinces of the United States.

As Lindgren[62] pointed out in 1902, the principal gold-deposits of the United States may be divided into four groups. The deposits of each group belong mainly to one metallogenetic epoch, and certain relationships are clearly shown. This classification, which has thrown much light on the genesis of the deposits, is useful as an instrument for study and for comparison of the deposits with respect to the problem of the migration of gold in them.

1. The Appalachian gold-deposits, and those of the Homestake type in South Dakota, are the most important representatives of the oldest group. These deposits generally yield placers, are usually low grade below the water-level and are singularly free from bonanzas. They are, in general, not greatly leached near the surface, and may have been enriched by the removal of other material more rapidly than gold. At only one of them, the Haile mine, in South Carolina, it is thought probable that gold has been carried below the water-level. The Homestake mines show little evidence of secondary enrichment by transfer of gold, as will appear in the review that follows. Judging from descriptions, practically all of these deposits are free from manganese.

2. The California gold-veins and related deposits in Nevada (Silver Peak) and in Alaska (Treadwell, etc.) are younger than the Appalachian deposits, and were probably formed in the main in early Cretaceous times. These deposits, where physiographic conditions are favorable, have generally yielded rich placers. At many places, moreover, the ore is worked at the very surface, and, as will appear in the subsequent review, there is very little evidence of the migration of gold to the deeper zones. In the places where detailed work has been done, rhodochrosite is never a gangue-mineral, although manganese oxide does occur in traces in the country-rock, and rhodochrosite is found in a few places in veinlets in the mining-districts but not associated with the gold-veins.

3. The deposits of the third group are later than the early Cretaceous, and some of them are probably early Tertiary.

[62] The Gold Production of North America, *Trans.*, xxxiii., 790 to 845 (1903); Metallogenetic Epochs, *Economic Geology*, vol. iv., No. 5, pp. 409 to 420 (Aug., 1909).

They are extensively developed in Montana, Nevada, Utah, and Colorado. Mr. Lindgren calls this group the Central Belt. Many of its deposits have yielded considerable gold, and in certain other districts very closely related genetically (Butte, Georgetown silver-gold lodes, Cortez Nevada, Tintic, etc.) much gold has been obtained as a by-product to copper- or silver-mining. Some of these deposits have yielded placers and some have not. At Philipsburg and Neihart, Mont., Georgetown, Colo., and elsewhere, the deposits show a secondary enrichment of silver below the water-table. At Philipsburg, and probably at some other places, an enrichment in gold accompanies this concentration of silver. Some of the lodes of group 3 carry much manganese, and some carry none. Present data are meager for most of these districts. The determination of gold from the surface down in a large number of deposits would serve as a useful check to the conclusions based upon the chemistry of the processes involved in its solution and precipitation.

4. Group 4 includes the most recent ore-deposits in the United States. All of them are Tertiary, and most of them are Miocene or Pliocene. In general, they were formed relatively near the surface, and in some places it is highly probable that not more than a thousand feet of vein-material has been removed by erosion since the ores were deposited. The majority of these deposits carry silver, and in many of them its value is greater than that of the gold; but they have supplied, notwithstanding, about 25 per cent. of the gold-production of North America. They are typically developed in Nevada (Comstock, Tonopah, Goldfield, Tuscarora, Gold Circle); California (Bodie); Idaho (De Lamar); South Dakota (later than Homestake type); Colorado (Cripple Creek, Idaho Springs, Rosita Hills, San Juan, etc.); Montana (Little Rockies, Kendall, etc.). Many occurrences in Mexico should probably be placed here, and likewise those of the Aleutian Islands, described by Becker. The deposits of this group have not supplied much placer-gold. They have not been exposed to erosion so long as the older deposits. In general, the gold is finely divided. It may have been scattered or it may have been redissolved and deposited lower down, Many of these deposits are in arid countries, where conditions for working placers are

not favorable; but, even those in well-watered districts supply relatively little placer-gold. Manganese is abundant in some of these deposits (Comstock, Exposed Treasure, Tonopah); it is very sparingly present in others (Little Rockies); in still others (Goldfield) it is almost entirely absent.

A few small placers are associated with the manganiferous lodes, although at some places, as at Tuscarora, Nev., they seem to have been derived from veins near-by which are not manganiferous, as is probably the case with some deposits of group 3 (Butte, Philipsburg). Many of the California veins (group 2) carry rich ore at the very surface, but the Tertiary gold-veins are generally richer in gold a few feet below the surface than at the outcrop. Doubtless, many of them would have been overlooked if it had not been for the concentration of horn-silver and argentiferous pyromorphite at the surface. At many of these deposits, however, good gold-ore is found only a few feet below the surface.

It thus appears that practically all of the manganiferous gold-deposits of the United States, so far as they have been described, may be included in groups 3 and 4; that nearly all described deposits where relations indicate a migration of gold belong to the same groups; that placers are much less abundantly developed than in groups 1 and 2; and that outcrops less frequently supply gold; that secondary enrichment below the water-table, if carried on at all, proceeds with extreme slowness in groups 1 and 2, but may be more pronounced in deposits of groups 3 and 4. Not all of these deposits carry manganese, however, and those which do not carry it should be expected to show relationships more nearly approximating those of groups 1 and 2.

5. Some deposits formed at hot springs carry gold. As a rule, traces only are found in the sinters, and at many places even traces are not detected. This is readily explained when it is noted that these springs frequently carry both sulphates and iron. If the sulphates are due to contamination with oxygenated surface-waters, then such waters, before complete oxidation, would precipitate gold. Since only a little ferrous sulphate precipitates practically all of the gold in a solution, it would be supposed that the major deposition would be some distance below the surface, where oxygen-bearing waters first

contaminated the hot solutions, and not at the surface. The same argument should apply to silver also, although the action of the ferrous salt on solutions carrying silver is not nearly so rapid as on solutions carrying gold. In the hot solutions, manganese, even if it were present, would probably not hold the gold or silver in solution by oxidizing ferrous salts, for ascending hot waters deposit manganous rather than manganitic compounds.

1. *Southern Appalachian Districts.*—The gold-deposits of the southern Appalachians are among the oldest gold-deposits of the United States, and were probably formed[63] in the main, 3 or 4 miles below the surface at the time of deposition. Many of them are in mica-schist and other crystalline rocks, and some are closely associated with granitic intrusions. Some are cut by diabasic intrusives, presumably later than the ore. The deposits have yielded considerable placer- and lode-gold. The minerals, according to Graton,[64] include quartz, sericite, biotite, fluorite, gold, pyrite, galena, blende, pyrrhotite, chalcopyrite, magnetite, etc. Manganese-minerals are not mentioned. In Becker's tabulation of the minerals of the gold-mines of the southern Appalachians, compiled from all previous descriptions and including mines not described by Graton, pyrolusite is mentioned in only three mines and rhodochrosite in one.[65]

Few of these deposits have been extensively explored in depth, and consequently data respecting the vertical distribution of the gold-values are meager. Many of them are profitable near the surface, partly by reason of the rotten condition of the rock, which renders it more easily worked, and partly because gold is accumulated or enriched by the removal of valueless material. In general there is, according to Graton, very little evidence for or against the theory of the migration of gold; but such migration, if it has taken place, has been extremely slow, for areas which have probably been exposed since Tertiary time show a marked concentration at and near the surface. Possibly some gold has been transferred to lower

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[63] Lindgren, *Bulletin No.* 293, *U. S. Geological Survey*, p. 124 (1906).

[64] *Idem*, p. 62.

[65] *Sixteenth Annual Report*, *U. S. Geological Survey*, Part III., *Mineral Resources of the U. S.*, p. 277 (1894–95).

levels at the Haile mine, South Carolina,[66] where the limit of profitable mining is in general less than 200 ft. below the limit of complete oxidation. In this zone scales of pyrite and free gold are found in joint-cracks, indicating a comparatively recent age. The deposits are cut by basic dikes. Prior to Graton's work, many thought that the primary deposition of gold was genetically related to the dikes,[67] since the workable ore appears to be limited to the area cut by them. If the basic dikes (like most basic rocks) carry manganese, then our hypothesis supports, and is supported by, Graton's opinion that secondary enrichment has probably taken place, and the conflicting views of Graton and Maclaren respecting the genesis of the ores are thus reconciled.

Certain ore-deposits of Alabama recently described by H. D. McCaskey[68] comprise fissure-veins in granite and lenticular bodies in schists. The principal minerals are quartz, pyrite, and gold. Some garnet is found in the vein-quartz at Pinetuckey. Weathering extends to water-level (from 40 to 80 ft. below the surface). The ores are oxidized above this level and are generally free-milling, but below this level the ore is not profitably amalgamated so far as explored in depth. The ores are fairly regular in width and values, and no evidences of enrichment below the water-level are recorded.

2. *Black Hills, S. D.*—The principal gold-deposits of the Black Hills[69] are in pre-Cambrian schists which, like the ore-bodies, are cut by Tertiary intrusives. Since the Cambrian conglomerates contain placer-gold,[70] some of the ores must have been deposited in pre-Cambrian times. The most important deposits are comprised in the Homestake belt, about 3 miles long and 2,000 ft. wide. The principal minerals are quartz, dolomite, calcite, pyrite, arsenopyrite, and gold, with which are associated the minerals of the schist: quartz, orthoclase, hornblende, biotite, garnet, tremolite, actinolite, titanite, and graphite.[71] The ores, though uniformly of low grade, are

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[66] Graton, *Bulletin No.* 293, *U. S. Geological Survey*, p. 67.

[67] Maclaren, *Gold*, pp. 57, 592 (1908).

[68] *Bulletin No.* 340, *U. S. Geological Survey*. p. 36 (1908).

[69] Irving, Emmons, and Jaggar, *Professional Paper No.* 26, *U. S. Geological Survey* (1904).

[70] W. B. Devereux, *Trans.*, x., 469 (1881–82).

[71] J. D. Irving, *loc. cit.*, p. 90.

very profitable. Some of the ores at the surface were below the average tenor, while other surface-ores were two or three times as rich as the average. The values extend downward as far as exploration has gone, and are fairly uniform to 1,000 ft. or more below the surface. In general, according to S. F. Emmons, secondary enrichment by surface-leaching has had relatively small importance.[72]

3. *Treadwell Mines, Alaska.*—At the Treadwell mines, Douglas Island, Alaska, large dikes of albite-diorite intrude greenstones and schist, and the shattered diorite has been extensively replaced by mineralizing solutions, and cemented by low-grade gold-ore. The minerals include quartz, albite, rutile, chlorite, epidote, calcite, siderite, pyrite, pyrrhotite, magnetite, chalcopyrite, and molybdenite. Manganese-minerals are not reported.

The mines have been developed 2,000 ft. down the dip. According to A. C. Spencer,[73] the ore shows no progressive change in appearance or values with increasing depth. In the lowest level it is quite as rich as in the upper workings; and it is evident that changes on the dip are no greater than along the strike. Nothing in the character of the ore indicates any important concentration of values by oxidizing waters. The fact that extensive placers were not formed is not opposed to the view expressed by Spencer that the gold has not been transferred; the country has been recently glaciated, and surface-accumulations have been scattered. The gold accumulated at the apex since glacial time was, indeed, recovered by sluicing.

4. *Berner's Bay, Alaska.*—According to Adolph Knopf, the lodes of the Berner's Bay district are fissure-veins in diorite. There is no evidence of secondary enrichment of gold or of leaching near the surface. The deposits contain no manganese.

5. *The Mother Lode District, Cal.*—The Mother Lode district, as described by F. L. Ransome,[74] is an area of crystalline schists and altered igneous rocks with intruded granodiorite and related rocks. The deposits are fissure-veins, which generally trend northwestward, and, at many places, parallel the schistosity of the country-rock. The ore does not contain

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[72] *Professional Paper No. 26, U. S. Geological Survey*, p. 79 (1904).

[73] *Bulletin No.* 287, *U. S. Geological Survey*, pp. 32 and 115 (1906).

[74] Mother Lode District, *Folio No.* 63, *U. S. Geological Survey*, p. 3.

manganese-minerals. Placers are abundantly developed, and at many places rich ore is found at the very surface. According to Ransome, there is no evidence that the mines grow suddenly richer at any arbitrary depth, nor is there any recognizable regular change in the value of pay-shoots with depth, below the zone of superficial weathering. Some of these deposits are very regular and uniform in values, and have been developed to very great depth.

6. *Nevada City and Grass Valley, Cal.*—The area of Nevada City and Grass Valley [75] includes metamorphosed Carboniferous sedimentary rocks, compressed into isoclines, and associated igneous rocks less intensely metamorphosed. Above these are slates with associated diabase and serpentine. These rocks are folded and metamorphosed, but are not so intensely compressed as the Carboniferous. Intruded into these rocks are great bodies of granodiorite, probably of early Cretaceous age. The ore-deposits are strong fissure-veins, formed after the granodiorite intrusions. The minerals are quartz, chalcedony, magnetite, sericite, mariposite, pyrite, pyrrhotite, chalcopyrite, galena, blende, scheelite, arsenopyrite, tetrahedrite, stephanite, and cinnabar. Some earthy manganese-ore occurs in small fissures in the granodiorite, but not in connection with the quartz veins.

Near the surface [76] the upper part of a vein is generally decomposed, forming a mass of limonite and quartz. The decomposition seldom extends more than 200 ft. on the incline of a vein dipping 45°, or more than 150 ft. below the surface. Fresh ore is sometimes found almost at the surface. The surface-ore is generally richer than the fresh ore below, owing to the liberation of gold from the sulphides and the removal of substances other than gold. In this process, silver is also partly removed. In some of the mines, the lodes have been followed down the dip for 2,000 or even 3,000 ft. The unoxidized ore shows no gradual diminution of tenor in the payshoots below the zone of surface-decomposition. "Within the same shoot there may be many and great variations of the tenor, but there is certainly no gradual decrease of it from the

[75] Waldemar Lindgren, *Seventeenth Annual Report, U. S. Geological Survey*, Part II. (1895–96).

[76] *Loc. cit.*, p. 128.

surface down."[77] Important placer-deposits were formed from these veins.

7. *The Ophir District, Cal.*—The rocks of the Ophir district[78] comprise amphibolite-schists and massive amphibolites, with intrusions of granodiorite. These rocks are cut by quartz veins which fill co-ordinate fissures. The minerals are gold, electrum, some iron, copper and arsenical pyrites, with galena, blende, tetrahedrite, and molybdenite. The gangue is mainly quartz with a little calcite. The proportion of gold to silver varies by weight from 1 : 1 to 1 : 10, the values of gold predominating. Certain small ore-shoots, in veins in the amphibolite, carry more than the usual tenor of gold; and the richest shoots are usually found where veins cross the belts rich in iron. According to Lindgren, such ore-bodies may have been enriched by leaching. The common statement, that the gold-vein becomes barren as the depth from the surface increases, is not justified, in his opinion,[79] by the evidence afforded in the mines. The extensive development of placers, the value of the ore near the surface, and the occurrence of valuable ore-shoots just below the surface, are opposed to the notion of extensive migration of gold in these deposits.

8. *Silver Peak, Nev.*—According to J. E. Spurr,[80] the deposits of Silver Peak, Nev., are lenticular masses and fissure-veins in Palæozoic sedimentary rocks. Genetically, they are related very closely to granitic rocks, which, as shown by Mr. Spurr, have alaskitic or pegmatitic phases. They are probably post-Jurassic, and should be grouped with the California gold-veins, with which geologically they have much in common. Of the Drinkwater and Crowning Glory deposits, which are the most important examples, Spurr says that no decided enrichment of the ores by oxidation can be established. The ores in the upper tunnel seem to have been locally richer than any found in the lower tunnel; but this difference has no evident relation with the surface, and is probably original. The values are finely disseminated gold and auriferous sul-

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[77] *Op. cit*, p 163.

[78] Waldemar Lindgren, *Fourteenth Annual Report, U. S. Geological Survey*, Part II., p. 252 (1892–93).

[79] *Idem*, p. 279.

[80] *Professional Paper* No. 55, *U. S. Geological Survey* (1906).

phides, scattered through vitreous quartz. The character of the ore affords no ground for supposing any great concentration by surface-waters, since the minerals are not easily reached by percolating waters. No ore-shoots correspond to the fractures which cross the ore—an indication that the waters which circulated along such subsequent fractures had little effect in the redistribution of values.

9. *Philipsburg, Mont.*—The Philipsburg quadrangle is an area of sedimentary rocks, ranging from pre-Cambrian to late Cretaceous, with intrusions of quartz-monzonites and related rocks, probably belonging to the same period of intrusion as that of the Butte granites and other batholiths in Montana. The most important ore-deposits in this quadrangle are those of the Granite-Bimetallic and the Cable mines.

The Granite-Bimetallic is a strong fissure-vein in quartz-monzonite, which carries chiefly silver, but also an important amount of gold. There is conclusive paragenetic evidence of the secondary enrichment of silver below the water-level, and the rich silver-ore carries also more gold than the low-grade silver-ore in the bottom of the mine. The outcrop of this deposit carried some silver, but very little gold; and, after the discovery, the location was allowed to lapse, by reason of the small assay-returns from the gossan. Richer ore appeared not far below the surface and extended down to the 10th level. The shoot of high-grade ore, which extended for about a mile along the strike of the deposit, followed, in a broad way, the present rugged surface. The gangue is rich in manganese. Some migration of gold has undoubtedly taken place. No associated placers have been developed.

At the Cable mine the deposits are included in a long, thin block of limestone, in contact on either side with quartz-monzonite. The principal minerals are calcite, quartz, pyrrhotite, pyrite, magnetite, and chalcopyrite, with chlorite, muscovite, and other silicates. At one or two places small traces of manganese dioxide have been noted in the oxidized ore, but it is very much less abundant than in the deposits of the Granite-Bimetallic type. This deposit yielded important placers. Good ore was found at or very near the surface; and, according to the best obtainable data, the values increased somewhat for a short distance below the surface. Some concentration has taken

place by the removal of calcite and other valueless material more rapidly than gold; but there is no evidence of secondary enrichment in gold below the water-table. The indications are, that the gold has not been extensively transported since the deposit was formed.

10. *Other Montana Districts.*—The secondary enrichment of gold- and silver-deposits at Neihart, at Butte, and in other Montana districts, has been described by W. H. Weed in various papers. These deposits generally contain appreciable manganese. In that respect they differ from the Idaho deposits described by Lindgren, which do not carry rhodochrosite or appreciable manganese dioxide. With some notable exceptions, such as the De Lamar deposits, the Idaho veins are probably older than those of Montana, and, as Lindgren has pointed out, should be grouped with the early Cretaceous California gold-veins rather than with the late Cretaceous or early Tertiary group, to which most of the Montana deposits belong. The Idaho veins which have been closely studied do not give evidence of the downward migration of gold.

11. *Edgemont, Nev.*—The gold-deposits at Edgemont, Elko county, Nev., which should be classed with group 3, are in an area of quartzite, with intrusions of granodiorite. The deposits are fissure-veins, and their gold-values are comparatively uniform. The ore consists of pyrite, galena, and arsenopyrite in a gangue of quartz. Copper carbonates and manganese-minerals are rare or absent. The ore is stoped practically to the surface. There has probably been a slight amount of enrichment by removal of certain substances in the oxidized zone more rapidly than gold; there is no evidence that gold has been transferred below the water-level by descending surface-waters.[81]

12. *Leadville, Colo.*—The deposits of Leadville yield silver, lead, and gold. The country is an area of Palæozoic limestones and quartzites, with intrusive sills and dikes of porphyries.[82] Some of these deposits carry in the upper horizons a large amount of manganese; and this ore is frequently rich in silver, presumably in the form of the native metal or as chloride.

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[81] W. H. Emmons, *Bulletin No.* 408, *U. S. Geological Survey* (1910).

[82] S. F. Emmons, Geology and Mining Industry of Leadville, Colorado, *Monograph No. XII., U. S. Geological Survey* (1886); and S. F. Emmons and J. D. Irving, *Bulletin No.* 320, *U. S. Geological Survey* (1907).

Assays of this ore, showing the amount of gold contained in it, are not available to me. According to the requirements of the theory under investigation, it would be expected to be low in gold in this upper zone, where the waters probably carry ferric salts and chloride. Of considerable interest in this connection are some small fractures in the quartzite at a lower horizon, which, as Mr. Emmons informs me, often carry small amounts of high-grade manganiferous gold-ore. This ore he regards as a deposit from descending waters. Possibly it is the gold leached out above, where ferric salts predominate, and was carried to greater depth by the manganiferous solutions which delay the action of ferrous sulphate as a precipitant of gold.

13. *Georgetown, Colo., Silver-Lead Deposits.*—The silver-lead lodes of Georgetown and Silver Plume, Colo., are of early Tertiary age. The veins cut crystalline schists and Tertiary igneous rocks. According to Spurr, Garrey and Ball,[83] several thousand feet of overlying rocks have been eroded since the ores were deposited, and some of the values in the eroded portions of the lodes have migrated to the portions still remaining. The principal metallic minerals are argentiferous galena and blende, with pyrite and chalcopyrite; the ores usually carry about $2 gold per ton. The silver-values are mainly in polybasite, freibergite, argentite, pyrargyrite, and proustite. The gangue is quartz, chalcedony, barite, with carbonates of lime, iron, manganese, and magnesia.

The rich silver-minerals were the last to be deposited, and form on the walls of the fractures in the older, baser ore, or cut the older deposits. The zone of complete oxidation extends from 5 to 40 ft. below the surface. The oxidized ore often contains several hundred ounces of silver per ton. Below this ore are friable black sulphides and secondary galena. This secondary ore, according to Spurr and Garrey, is rich in silver and lead and carries more gold than occurs at greater depth. This ore cuts the primary sulphide; and the latter, which may have contained from 20 to 30 oz. of silver per ton, is enriched to more than 200 oz. per ton.

Quoting from Spurr and Garrey[84]:

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[83] *Professional Paper No* 63, *U. S. Geological Survey,* p. 136 (1908).

[84] J. E. Spurr and G. H. Garrey, *Professional Paper No.* 63, *U. S. Geological Survey,* p. 144 (1908).

"Below the zone where soft secondary sulphides occur and irregularly overlapping the lower portion of this zone the rich ores contain polybasite, argentiferous tetrahedrite, and ruby silver, better crystallized and more massive than the pulverulent sulphides, but also subsequent in origin to the massive galena-blende ore. These richer ores diminish in quantity as depth increases, though gradually and irregularly, so that the lower portion of the veins contains relatively less silver and lead. The best ore in most veins has been found in the uppermost 500 feet, although good ore extends locally down to 700 or 800 feet, and in the Colorado Central, and to a minor extent in other veins, down to a thousand feet or more."

14. *Auriferous Deposits of the Georgetown Quadrangle, Colorado.*—The auriferous deposits of the Georgetown quadrangle are mainly at Idaho Springs and in the Empire district, although some are developed near Georgetown in the area of the silver-lead deposits. As shown by Spurr and Garrey, the gold-lodes are probably of later age than the silver-lead deposits. They cut the crystalline schists and the Tertiary porphyries, but are genetically related to alkali-rich intrusive rocks of middle or late Tertiary age. They carry pyrite, chalcopyrite, chalcocite, quartz, adularia, and gold, with minor amounts of barite, fluorite, telluride, etc. Carbonates of iron, magnesium, lime, and manganese occur, but are relatively rare. In many of the mines the ore averages from 1 to 2 oz. of gold and from 20 to 40 oz. of silver per ton. The lodes are usually oxidized at the surface and from 15 to 70 ft. downward. They have yielded some moderately-productive placers. In several mines, the oxidized is much richer than the average ore. Below the zone of oxidation, secondary chalcopyrite and chalcocite prevail for several hundred feet from the surface, but decrease at greater depth. There is an important enrichment of gold and silver, coincident with the occurrence of the copper-minerals. As stated by Spurr and Garrey:

"In the mines mentioned a portion of the copper which has contributed to the enrichment of the original sulphides has been derived from the oxidized zone, but it seems unlikely that this has been the case with the gold and silver, which, like the enriched superficial portions of the argentiferous veins, must have been derived from the overlying portions of the lodes which are now eroded. . . . .

"On the whole, the strongest evidence of the reworking of the ores by surface waters is afforded by markedly cupriferous ores . . . Apart from this, however, and from the probable partial concentration of galena near the surface in some mines, the evidence of rearrangement of the ores by descending waters is in general not nearly so great as in the Georgetown district, and such reworking has probably taken place to a considerably less extent."

15. *San Juan, Colo.*—The gold-deposits of the San Juan region,[85] including those near Telluride, Silverton, and Ouray, are, as shown by Ransome, of varied character. They are mainly Tertiary, and should be classed with group 3 or 4 above named. The lead-silver deposits and the stocks near Ironton are not here considered.

In this elevated area the ground is frozen much of the year, and the rapid erosion is due largely to mechanical disintegration. Secular decay or oxidation of the ores, according to Ransome, is not as a rule very extensive, and is at some places negligible. Purington has pointed out, however, that the outcrops of the San Juan lodes are, in general, of lower grade than the ore a few feet below the surface, possibly by reason of the migration of gold in suspension. Many of the lodes are tight, and do not appear to offer favorable conditions for downward migration of waters. The country is well drained, and chlorine is probably not abundant in the mine-waters. The conditions for deep-seated enrichment are therefore not particularly favorable, although some concentration has taken place locally by the leaching and removal of the less-valuable materials from the ore. The workable ore appears to be mainly of primary origin.

At some places the gangue includes manganiferous minerals. There is some evidence that gold was transported to a limited extent. As Ransome points out,[86] in the Tomboy and Camp Bird mines, black oxide of manganese occurs in the deepest workings (in 1901) and usually indicates good ore. These little sheets of rich, dark, manganiferous ore, which fill post-mineral fractures, Ransome regards as later than the general mass of the ore. It is reasonable to suppose that they represent the deposition from solutions which dissolved gold in the upper portion of the lode, where ferric salts prevail, and which, in the presence of manganese, were able to transport their load to greater depths, but which, coming into contact with pyrite,

[85] F. L. Ransome, A Report on the Economic Geology of the Silverton Quadrangle, *Bulletin No.* 182, *U. S. Geological Survey,* (1901); and C. W. Purington, Preliminary Report on the Mining Industries of the Telluride Quadrangle, *Eighteenth Annual Report, U. S. Geological Survey,* Part III., p. 745 (1896–97); Purington, Woods, and Doveton, The Camp Bird Mine, Ouray, Colo., *Trans.*, xxxiii., 499 to 550 (1903).

[86] *Op. cit.*, p. 101.

were ultimately reduced and forced to give up their gold when, through the oxidation of pyrite, ferrous sulphate had been formed.

16. *Cripple Creek, Colo.*—The gold-deposits of Cripple Creek, Colo., have yielded some $200,000,000 gold and less than $1,000,000 silver. The lodes are fissure-veins and replacement-deposits in volcanic breccia, in Tertiary intrusive rocks, and in granite. The fissures, according to Lindgren and Ransome,[87] were formed at about the same time as the intrusion of associated basic dikes, and represent a late phase of volcanic activity. The deposits are probably of middle or late Tertiary age, and were formed by hot ascending waters, relatively near the surface. Calaverite is the chief primary constituent; native gold is rarely present in the unoxidized ores. Pyrite is widely distributed; tetrahedrite, stibnite, and molybdenite are sparingly present. The gangue is quartz, fluorite, adularia, carbonates (including rhodochrosite), some sulphates, etc. Some of the deposits were workable at the surface, but the placers which have formed are relatively unimportant. Although rhodochrosite is subordinate in amount, the highly-fractured country-rock contains appreciable manganese (0.20 ± per cent.). According to Lindgren and Ransome, the processes of oxidation were attended by the formation of kaolin, hydrous silica, and oxides of iron and manganese. Manganese oxides are often present in the oxidized zone, and, according to Penrose, form nodules in the Pharmacist and Summit mines. They result from the alteration of rhodochrosite, manganiferous calcite, or other minerals, and are generally distributed in the oxidized zone as stains filling cracks and fissures.[88] During oxidation, manganese is greatly concentrated in the seams of the rock. In general, the lower part of the zone of oxidation is above water-level, and usually less than 200 ft. below the surface. In some places silver has been completely leached from the oxidized ores. Horn-silver is not noted.

Whether a slight enrichment of gold has taken place in the oxidized zone it is not easy to decide. Lindgren and Ransome are inclined to the belief that the oxidized zone as a whole

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[87] *Professional Paper No.* 54, *U. S. Geological Survey* (1906).

[88] *Idem*, p. 123.

is somewhat richer than the corresponding telluride zone.[89] If this is true, no extensive downward migration of gold can have taken place. The trivial enrichment in the oxidized zone may have resulted from the removal of some constituents of the primary ore.

If gold was dissolved in the Cripple Creek deposits, it was precipitated again at practically the same horizon; for, in these deposits, the zone in which solution takes place is as rich or richer than that in which precipitation usually takes place. The ground is open, providing paths for downward-circulating waters, but it should be remembered that, while the ore-bearing complex is very pervious to water, it is surrounded by impervious rocks. After the volcanic rocks had been drained in mining, the flow of water was comparatively small. Lindgren and Ransome have compared the volcanic complex to a "sponge in a cup." As shown by them, the conditions for a circulation of atmospheric water were most unfavorable—a fact which had an important bearing on their conclusion that the ores had been formed by magmatic waters. In the absence of a circulation, the gold could not be transported. A check to this reasoning with respect to a downward circulation is the fact that in the porous, brecciated mass, filled with stagnant water, the oxidation extended downward to a depth generally less than 200 ft., and even in this zone residual sulphides are often present. If the solutions did not carry oxygen downward, it would be supposed that they could not carry gold; and if the latter had been dissolved at the higher levels, in the absence of a circulation it could not descend. There is some evidence which may be interpreted as an indication that the gold migrated laterally, or possibly that it has been precipitated essentially in place from cold solution. Richard Pearce[90] has recorded analyses of oxidized and unoxidized ore. The material for the analyses was taken from a section drawn clear across the two different portions of the specimen. The analyses show that the oxidized ore carries 14.58 oz. of gold per ton, or 2.34 oz. more gold than the unoxidized ore, and that all the silver has been leached out. In ore so rich such a con-

[89] *Professional Paper No. 54, U. S. Geological Survey,* p. 203 (1906).

[90] Further Notes on Cripple Creek District, *Proceedings of the Colorado Scientific Society,* vol. iv., pp. 11 to 16 (1894–96).

centration may result merely from leaching-out of the substances other than gold; but, on the other hand, the analyses of the altered rock indicate that little leaching of the silicate minerals has taken place, and that the oxidized portion was originally richer than the unoxidized, or else that some gold had been added. Since 0.27 per cent. of MnO_2 is present in the oxidized ore, while none is reported in the unoxidized ore, it appears that MnO_2 was added in the process of secondary alteration, and it is possible that the same solutions added gold and iron.

If the gold was dissolved in the Cripple Creek "sponge," it was precipitated in the stagnant solutions where they were in contact with pyrite. In the absence of a downward circulation of water, such lateral migration would not be unlikely.

The results of oxidation processes are described as follows:[91]

"Thorough oxidizing decomposition will destroy the original structure of this vein. In sheeted lodes with many small parallel fissures and joints the latter may become effaced and the lode appears as a homogeneous brown, soft mass. In other cases a central seam may be retained and usually appears as a streak of soft, more or less impure kaolin; in other cases it may be filled by white compact alunite, more rarely by jasperoid or opaline silica. Crusts of comb quartz, if originally present, lie included in the clayey seams, but neither the original fluorite nor the carbonates are ordinarily preserved. Very rich oxidized ore sometimes fills the central cavities of the lode like a thick brown mud of limonite, kaolin, and quartz sand, and easily flows out when the vein is opened."

It should not be inferred, however, where channels are large and open that the rich, gold-bearing brown mud is necessarily a deposit from solution. It may have been carried down in suspension; for similar rich mud, with 2 oz. of gold per ton, was found on the floor of the 12th level of the Gold Coin mine after it had been filled with water and allowed to stand.

It thus appears that the conditions at Cripple Creek, which at first appear fatal to the hypothesis, may be rationally explained, when it is recalled that downward migration of gold depends not only upon solution and precipitation, but requires a circulation, and that conditions for a circulation here were peculiarly unfavorable. They show also that conditions for a relatively rapid circulation are prerequisite, if the dissolved gold is to be carried below the zone of mixed oxides and sulphides.

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[91] *Professional Paper No. 54, U. S. Geological Survey,* p. 199 (1906).

17. *Summit District, Colo.*—This district is located southwest of Alamosa near the Rio Grande-Conejos county-line. According to R. C. Hills,[92] the metal-bearing horizon is near the middle of the Tertiary eruptive series of south and southwest Colorado. The associated rocks are andesites, trachytes, rhyolites, etc.; but, unlike the eruptives of most Tertiary districts in this province, these rocks appear to have been closely compressed, yielding a series which, as shown in Mr. Hills's sketches, are probably isoclinals. Some features of the ore-deposits are puzzling; but, whatever their genesis, they illustrate very clearly the theory of secondary enrichment—a fact which was fully recognized by Mr. Hills as long ago as 1883.

The ore-bodies, so far as exposed, are rudely tabular and approximately vertical. The ore is chiefly quartz and pyrite, but contains some enargite, galena, sphalerite, and other minerals. Placers appear to be of subordinate importance. The mineralized matter may be separated into three divisions: (1) the impoverished zone near the apex; (2) the zone of rich and partly-oxidized ore; and (3) the low-grade sulphides. The zone of impoverishment, with two exceptions, includes the outcrops of all the lodes and extends downward to 50 ft. or more. The zone of incompletely-oxidized ore extends to a depth varying from a few feet to 300 ft. In this zone the quartz is colored dark brown by oxides, and the more-highly auriferous material is characterized by an abundance of brown oxide. The gold in this ore carries only about 0.025 silver. All the bonanzas were, according to Mr. Hills, confined to this zone. In some places gold appears in a disseminated form in innumerable small grains, so aggregated as to resemble a continuous sheet of metal. Locally, the grains unite and form flat nuggets, one or more ounces in weight. The occurrence of this richer material is confined, according to Mr. Hills, to the immediate vicinity of a central channel which has been filled with earthy matter, fragments of rock and iron oxides. Some of the rich seams of gold powder have been introduced into fractures which cut barite. Below the rich and partly-oxidized ore, the primary sulphides appear to have been unworkable under conditions then existing. There is, however, in three

[92] *Proceedings of the Colorado Scientific Society*, vol. i., p. 20 (1883–84).

mines[93] a concentration of silver at greater depth than that of the gold-bonanzas. Mr. Hills ascribes the two rich out-cropping ore-bodies, which are exceptional in this district, to intense kaolinization on either side of the ore-bodies, causing the country-rock to be much more readily eroded than the extremely hard quartz outcrop. This consequently remained considerably above the general surface, forming a precipitous ridge, which was, as he explains, protected from solution, which went on more vigorously below, in the places where snow and water accumulated.

Although Mr. Hills mentions brown oxides at several places, he does not say that they are manganiferous.

Dr. Raymond[94] says that the oxides include those of purplish hue.

18. *Bodie, Cal.*—The deposits of Bodie, Cal., are east of the Sierras, near the State-line. They are not of the California type, but are associated with andesite and belong to the late Tertiary group so extensively developed in Nevada. R. P. McLaughlin[95] has described the most important mines. The lodes are fissure-veins in andesite. Nearly all strike northward and are approximately parallel. The ore carries about equal amounts of gold and silver. The deposits are developed extensively to a depth of 500 ft. below the surface. One shaft is 1,000 ft., another 1,200 ft. deep. Outcrops of encouraging value are rare. Almost without exception the veins have failed to carry pay-ore beyond 500 ft. below the surface; but above this depth occur large, rich ore-bodies, which, according to McLaughlin, carry ore worth as high as $400 a ton. Faulting and displacement are probably of later date than the period of vein-formation. Some of the oxidized ore carries manganese dioxide. It is "loose and clayey in texture and carries some silver to the exclusion of gold."[96]

19. *Exposed Treasure Mine, Cal.*—The Exposed Treasure mine,[97] which is near Mojave, has produced considerable gold and silver. It is in an area of granitic rocks cut by quartz-por-

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[93] *Op. cit.*, p. 35.

[94] *Mines and Mining West of the Rocky Mountains*, vol. x., p. 329 (1875).

[95] *Mining and Scientific Press*, vol. xciv., No. 25, p. 796 (June 22, 1907).

[96] *Idem*, p. 796.

[97] Courtenay De Kalb, *Trans.*, xxxviii., 310 to 320 (1908).

phyry and capped by rhyolite. The lodes are probably Tertiary (group 4). The Exposed Treasure vein dips about 45° E. and is a sheeted brecciated zone. Considerable fissuring has taken place since the ore was deposited.

". . . . While the lodes are continuous, and often of great width, sometimes being 40 ft. and more from wall to wall, the pay-streaks, from 4 to 15 ft. in width, lie in well-defined chutes and overlapping sheets or lenses. It is noteworthy that only those chutes or lenses which now reach the surface contained important quantities of calcite and manganese dioxide."

The oxidized ores contain much MnO_2, the concentrates carrying 12 per cent. In the altered oxidized ore are kernels of ore containing pyrite, chalcopyrite, galena, and sphalerite, and these are richer in the precious metals than the altered friable ore. As observed by De Kalb:

". . . . The altered ore bore manifest signs of extensive leaching, and where it had become almost completely decolorized by the removal of iron, the precious metal contents had nearly disappeared, and such ore never contained copper except in the form of chrysocolla.

"The absence of sulphides in all the [oxidized] ores, except in the cherty skeletons, and in the undecomposed kernels of hard ore, was very complete. The mill-concentrates (150 into 1) had an average composition of SiO_2, 30; FeO, 37 . . . . and MnO_2, 12 per cent. These concentrates never contained more than 1.5 per cent. of sulphur.

"In the lower friable siliceous ores, the ratio of gold to silver was as 1 to 12, while in the upper mangano-calcitic ores the ratio was as 1 to 72. Assays of gold-scale, and of coarse gold panned out, from all parts of the mine, showed a remarkably uniform alloy of 1 part of gold to 0.461 part of silver. The silver in the upper portion of the mine was present almost wholly in the form of silver chloride.

"On the assumption, from the evidence, that the abundance of chlorides would prevent the leaching-out of the silver and its reconcentration below water-level, and that the ferric and cupric sulphates would have abstracted large quantities of the gold, which would be re-deposited lower down together with the copper in the form of secondary enrichments, it was natural to predict an ore below permanent water rich in these metals, and relatively lean in silver. It would be difficult to conceive a nicer justification of theory than that which was afforded when development at length extended below water-level. The ore consisted of a hard bluish-gray mass of original chert-cemented breccia, re-cemented by quartz, with partial replacement of the granite and quartz-porphyry by silica, heavily impregnated with sulphides, among which were considerable quantities of chalcopyrite, bornite, and some covellite. The gold-content of the ore had increased 150 per cent. above the average in the friable siliceous ores on the upper levels, and the ratio of the gold to silver was as 1 to 2."

20. *Tonopah, Nev.*—The deposits at Tonopah, Nev., are silver-gold replacement-veins in andesite. They are of mid-

dle or late Tertiary age, but possibly somewhat older than the Comstock lode. Placers are not developed. The primary ore, according to J. E. Spurr,[98] is composed of quartz, adularia, sericite, carbonates of lime, magnesia, iron, and manganese, with argentite, stephanite, polybasite, chalcopyrite, pyrite, galena, blende, silver selenide, and gold in an undetermined form. The zone of oxidation extends to greater depth in the more-highly fractured places; and for this reason the brittle and more-broken lodes are more-deeply oxidized than the wall-rock. The Mizpah vein is for the most part oxidized to a depth of 700 ft. Standing ground-water is lacking. The oxidized ore contains limonite and manganese dioxide, with plentiful horn-silver and some bromides and iodides of silver. The so-called oxidized ore from the outcrop down is, according to Spurr, a mixture of original sulphides (and selenides), together with secondary sulphides, chlorides, and oxides. At a depth of 500 ft. (in the Montana Tonopah mine) good crystals of argentite, polybasite, and chalcopyrite have been formed freely in cracks and druses of the sulphide ore. These minerals are later than the massive ore; but it cannot be shown that they were not deposited upon it by ascending waters. The case of dark ruby-silver (pyrargyrite) is different, however, for this is formed in cracks in the oxidized ore, and some argentite fringes minute particles of horn-silver as if secondary to it. "The evidence therefore favors the view that these secondary sulphides in the oxidized zone originated from descending surface waters, and probably part, but not all, of the sulphides in druses in the sulphide ore have a similar origin."

The waters which descend through the oxidized zone carry sulphates and chlorides, and "wad" is plentiful; but judging from the fairly-constant proportion of gold to silver (about 1 to 100 by weight) there has been little selective migration of gold and silver during oxidation, although the vein has been enriched to some degree by downward penetration of minerals leached from the outcrop as it was eroded. The rich ore-shoots, though partly oxidized, seem to be in the main original without thorough rearrangement. According to Mr. Spurr, this may be ascribed in part to the relatively scanty supply of water in this arid region.

[98] *Professional Paper No. 42, U. S. Geological Survey.* p. 90 (1905).

21. *Goldfield, Nev.*—The ledges of Goldfield are in middle or late Tertiary rocks, and, according to F. L. Ransome, were probably deposited within 1,000 ft. of the surface at the time of deposition. Ransome states convincingly the hypothesis that these deposits were formed by hot ascending solutions which mingled with descending sulphate-water contaminated by the oxygen of the air. Although the deposits are probably the most remarkable bonanzas of native gold-ores carrying little silver which have yet been discovered, it does not appear that they have been enriched to any considerable extent since they were deposited, for, as remarked by Ransome, it is difficult to harmonize the extent and intensity of alunitization which accompanies the gold with the hypothesis of the oxidation and enrichment of lean deposits during erosion. The mine-waters are rich in sulphates; and, judging from the geographical position of the deposits, they probably carry chlorides. Manganese dioxide is practically unknown in these ores, which in this respect differ from the ores at Tonopah and from a great many Tertiary deposits of the Great Basin province. No workable placer-deposits have been discovered; yet notwithstanding the fact that there may have been several hundred feet of vein-matter removed from these deposits since they were formed, there is little reason to suppose that much gold has migrated into the existing bonanzas from above. The gold is very finely divided, and could easily have been scattered, if it had been eroded with the ledges. As shown by the analyses of deposits elsewhere that were formed close to the surface by ascending hot waters, they seldom carry much gold. The maximum deposition is lower down; for, as soon as the ascending hot waters are contaminated by ferrous sulphate from the surface, gold must be precipitated.

The evidence offered at Goldfield is not out of harmony with the conclusion that, in the absence of manganese, gold is not readily transported in mine-waters.

22. *Manhattan, Nev.*—The gold-deposits at Manhattan, although inclosed in schists, are in an area of Tertiary volcanic activity, and should be classed with the deposits formed in Tertiary times. Although the schists contain stringers of gold of uncertain genesis, the principal deposits are steeply-dipping lodes of quartz and calcite, stained with iron and manganese

oxides. Some placers are developed. Rich ore was found very near the surface, but it was richer a few feet below the outcrop than at the surface. Some fracturing has taken place since the deposits were formed. In many instances the gold of the pockets of rich ore is intimately associated with iron and manganese oxides.[99] In view of the fact that the unaltered sulphides had not been encountered when the mines were visited, the character of the primary ore is unknown to me.

23. *Annie Laurie Mine, Utah.*—The Annie Laurie mine,[100] 175 miles south of Salt Lake, is in an area of dacite, rhyolite and rhyolite-tuff, and probably belongs to the later Tertiary group. The vein is poorly exposed at the surface, being largely covered by morainal material. Mr. Lindgren says:

> "The quartz forms an almost continuous sheet along the vein, rarely less than 3 feet in thickness and often expanding to a width of 20 feet or more. As a rule the walls are poorly defined and slickensides indicating motion are rare. In places it contains, parallel to the walls, streaks of iron oxides and black, sooty, manganese ores. . . .
>
> "The mine-workings have not penetrated below the zone of oxidation, and neither the quartz nor the country-rock seem to contain any unoxidized sulphides."

In the absence of extensive post-mineral fracturing, one would suppose that the conditions for migration of gold were not particularly favorable. Since the workings had not penetrated sulphide ore at the date of Lindgren's report, direct evidence was lacking.

24. *The Bullfrog District, Nev.*—In the Bullfrog district[101] the principal deposits are fissure-veins in rhyolite. The minerals include pyrite, quartz, and manganiferous calcite. Enough manganese is present in the calcite to stain much of the oxidized ore chocolate-brown or black. No placers are developed. The outcrops were comparatively poor, but within a few feet of the surface good ore was encountered, and some of the deposits were worked by open-cut. Some of the ore-deposits decrease in value below the 400-ft. level, where ore carrying

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[99] G. H. Garrey and W. H. Emmons, *Bulletin No.* 303, *U. S. Geological Survey*, pp. 84 to 93 (1907).

[100] Waldemar Lindgren, *Bulletin No.* 285, *U. S. Geological Survey*, pp. 87 to 90 (1906).

[101] Ransome, Emmons, and Garrey, *Bulletin No.* 407, *U. S. Geological Survey* (1910).

less than $5 per ton is encountered. Since the ore above this level carried many times this value, it appears that there has been a secondary concentration by surface-waters, and that the rich ore is related to the present topographic surface.

25. *Gold Circle, Nev.*—The deposits of Midas, Gold Circle [102] district, are in an area of late Tertiary rhyolites. The lodes are replacement-veins and sheeted zones and carry considerably more gold than silver (value). In the oxidized zone some of the ore is rich, but the sulphides are comparatively regular in value and give no evidence of extensive secondary enrichment. Some oxidized ore-shoots appear to have been increased in value by the removal of substances more soluble than gold. The minerals are chiefly quartz and pyrite. In the oxidized zone are seams of very rich gold-ore, composed of manganese, limonite, kaolin, and soft hydrous silica.

26. *Delamar Mine, Nev.*—The Delamar mine, in southeastern Nevada, is in quartzite cut by porphyry dikes of acid composition. It is presumably a Tertiary deposit, and is provisionally classed with group 4. The ore-body described by S. F. Emmons [103] is related to a strong zone of fracturing which strikes with the quartzite, but dips about 75°, or nearly at right angles to the dip of the quartzite. The ore is in shoots or zones of crushed quartzite. The chief ore-body, which is, roughly speaking, a long and comparatively thin, nearly upright cylinder, is divided into four parts by a dike of quartz-porphyry and a more basic dike, which cross nearly at right angles in the ore-body. The ore follows the line of intersection of the two dikes rather closely. The ore at the bottom of the mine consists of quartz and pyrite, which fill fractures in the altered quartzite. Where the dikes cross in the ore-body the light-colored dike appears to be continuous, but notwithstanding this the line of the dark dike across the light one is generally marked by a slight stain of manganese dioxide, which, as stated by Mr. Emmons, is characteristic of the "black" dike, and perhaps gives it that name.

Oxidation extends as far down as the tenth level. The ore that has been found below that level is too low in grade to pay

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[102] W. H. Emmons, *Bulletin No.* 408, *U. S. Geological Survey* (1910).
[103] *Trans.*, xxxi., 658 to 675 (1901).

for mining. The gold-ore carries silver and some copper. The tenor in gold increased from the surface downward to about the 7th level, although the values were not evenly distributed. Some lots of ore ran as high as 30 oz. per ton, and the richer parts of the mine averaged from $30 to $70 per ton. At the 10th level they had decreased to $4 or $5 per ton.

The Iron-Ore Deposits of the Moa District, Oriente Province, Island of Cuba.

BY JENNINGS S. COX, JR., SANTIAGO DE CUBA, CUBA.

(Wilkes-Barre Meeting, June, 1911.)

THE following notes, prepared in 1908, as the result of a personal examination and extensive explorations under my direction in 1906, have been revised and greatly augmented after two subsequent visits and further explorations in 1910.

The hard iron-ores of the south coast of Oriente Province, in the island of Cuba, have been known for many years, in fact one mine has been in operation for more than 20 years; but it is only within the past four years that the large mantle-deposits of brown ores of the north coast have attracted serious attention. A. C. Spencer[1] has described these ores in a paper, entitled Three Deposits of Iron Ore in Cuba, which refers to the Mayari, Moa, and Cubitas deposits. C. M. Weld,[2] in a scholarly paper, The Residual Brown Iron-Ores of Cuba, has discussed the character and possible genesis of these deposits with particular reference to the Moa field, calling attention also to the Taco and Navas fields in addition to those described by Spencer. Descriptive articles with reference to the Mayari field, and a few words about the Moa deposit, have also appeared.[3] It is the purpose of the present paper to give some account of the exploration of the Moa deposit and to call attention to its commercial importance.

LOCATION.

The Moa district is situated on the north coast of the Province of Oriente, which is the most easterly province of the

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[1] *Bulletin No.* 340, *U. S. Geological Survey,* pp. 318 to 329 (1908).

[2] *Trans.*, xl., 299 to 312 (1910).

[3] *Iron Age,* vol. lxxx., No. 7, pp. 421 to 426 (Aug. 15, 1907), and vol. lxxxi., No. 15, pp. 1149 to 1157 (April 9, 1908).

island of Cuba. It lies on the northern or seaward slope of the mountain range which, under various names, follows the north coast-line of the island, as a series of disconnected hills in the four central provinces and as a bold and continuous range, that forms a distinguishing feature of the landscape, in the most westerly province of Pinar del Rio and the most easterly province of Oriente. Moa lies about 35 miles west of the town and harbor of Baracoa, and nearly 45 miles east of the spacious harbor of Nipe. It is included in the Municipal District of Baracoa, in a sub-division thereof known as the "Barrio" of Nibujon.

The district is unsettled except for a few fishermen's huts near the coast. The coast-road from Mayari (Nipe bay) through Sagua de Tanamo to Baracoa passes through Moa, but this road is little traveled, amounts to scarcely more than a trail in some places, and is practicably impassable in the rainy season. The readiest means of access to Moa is by sea from Baracoa or Nipe bay. Baracoa is reached from Havana in three days, or from Santiago de Cuba in one day, by the Herrera Line, locally known as the "north-coast steamers." Nipe bay may be reached by rail from Havana in 24 hr., or from Santiago de Cuba in 6 hr., or by the north-coast steamers from the same ports. The Royal Mail Steam Packet Co. and the Munson Line run passenger-steamers from New York to Nipe bay direct. Tugs and small sail-boats are available at Baracoa or Nipe, but those at the latter place are larger and better. The trip by sea is more comfortable and quicker than overland. Under ordinary conditions, the best way to get to Moa is, therefore, by sea from Nipe bay. The trip must, however, be planned beforehand in order that the tug may get the necessary Custom-house clearance, otherwise it would be necessary to call at Baracoa first, to enter at the Custom-house before going to Moa, and time would be lost unnecessarily. As Moa is under the port of Baracoa, boats can sail from Baracoa direct to Moa.

Geology.

The underlying rocks of the island's structure are syenites, diorites, serpentines, and basalts, above which in many places is found a sheet of organic limestone, deposited previous to the upheaval by which the island emerged from the sea.

Highly characteristic of the north coast, and particularly of the northern slope of the north-coast range, is serpentine, which, in many places, is covered by a highly-ferruginous mantle, the product of its own decomposition.

In certain cases where local conditions have favored, nearly all the silica- and magnesia-contents have been carried off in solution and the residue is so high in iron as to attain the dignity of an iron-ore. It still carries, however, small quantities of silica, with the alumina and nearly all the chromium and nickel present in the original serpentine.

The following typical analyses first suggested the theory of the genesis of these deposits:

	Serpentine. Per Cent	Ore. Per Cent
SiO_2	39.80	6 26
MgO	33.69	
FeO	9.10	
Fe	......	45.67
Al_2O_3	1 39	10.64
Cr	0.20	1 96
Ni	0.94	0.845
Co	0.03	
P	......	0.008
S	......	0.107
Combined H_2O	......	11.59

Once one has studied the ore-body on the ground, a comparison of these two analyses carries the conviction that the serpentine is the parent of the iron-ore and that the latter is a residual deposit resulting from the decomposition of the former.

This theory has been independently arrived at and ably sustained by Weld in the paper already referred to, and, in 1910, Dr. C. K. Leith, after a careful study of the Mayari and Moa deposits, and analyses of the ore, foot by foot from surface to bed-rock, as well as of the rock itself, demonstrated beyond any reasonable doubt the correctness of the theory. This comment refers particularly to the four districts of the north coast of Oriente, since the Cubitas deposit differs from the others in certain essentials which would indicate a similarity but not an identity of origin.

The process of laterization by which these ores have been formed is typical of tropical regions, where the intense heat

and the abundant precipitation of moisture appear to carry the ordinary process of rock-disintegration and decay somewhat further than in more-temperate climates. Where ordinarily in the process of such disintegration the silica remains in the residue with the alumina and iron, laterization is characterized by the removal of nearly all the silica in addition to the more-soluble elements. Just what the chemistry of this process of solution of the silica is, has never, so far as I am aware, been definitely determined. Weld[4] quotes the ingenious theory of Sir Thomas Holland, Director of the Geological Survey of India, who suggests that laterization is due to the agency of lower organisms possibly akin to the so-called nitrifying bacteria.

He says:

> "With these are probably forms akin to the bacteria which oxidize and fix ferrous compounds, and which, precipitating the silica in the colloid form, permit its removal by the dilute alkaline solutions simultaneously formed."

Any detailed discussion of the facts and conditions which confirm this theory of the genesis of these ore-deposits is beyond the scope of this paper, and has, moreover, been rendered superfluous by the work of Weld and that of Leith, to which reference has already been made.

The red ferruginous mantle, product of the process of laterization, is typical of Pinar del Rio Province, and occurs in the northern part of Camaguey and Oriente Provinces, but so far as is known, with the exception of one place in Pinar del Rio and the Cubitas district in Camaguey Province, it is only on the north coast of Oriente, in the Mayari, Moa, Taco, and Navas districts, that conditions have resulted in residual deposits sufficiently high in iron to be classified as ores.

One deposit which I examined in Pinar del Rio, containing 50,000,000 tons of ore, and the immense deposit, exceeding 600,000,000 tons, at Mayari, are both characterized by a growth of pine timber and an almost complete absence of any undergrowth other than ferns; but this is not an essential characteristic, since at Moa some portions are covered with pine timber and others with various native trees and a dense jungle of undergrowth.

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[4] *Trans.*, xl., 308 (1910).

Smaller ore-bodies and near-ore bodies are reported from other provinces, but nothing has yet been examined to compare with the great deposits of Oriente Province, of which that of Moa is probably the most extensive.

Character of the Ore.

As indicated above, the ore consists of a mantle or blanket layer of varying thickness on the serpentine rock. The depth varies from nothing to more than 80 ft.; but, with the exception of one extensive bank in the Moa district, where there is a depth of from 50 to 80 ft., the average depth is fairly regular, running about 18 ft. This also checks very closely with the average depth of the Mayari deposit, the only one where thorough explorations are available for comparison.

The ore appears in three characteristic forms.

1. The great bulk of the deposit is an earthy mass, dark red to yellow in color, the yellow portions generally near the bed-rock and the darker red at the surface. Color is, however, no indication of the iron-content, but is apparently a function of the relative proportions of limonite and hematite in the ore. Near the surface the hematite predominates, and, with it, the red color, while close to bed-rock the iron is practically all present in the form of limonite, and the characteristic color is yellow. This condition has been carefully studied at Mayari, where the Spanish-American Iron Co. has for more than a year been engaged in mining this type of ore, and the conditions noted in these open workings are confirmed by borings made at Moa.

Charles Verlain, Professor of Geology at the University of La Sorbonne, Paris, in an article[5] elaborating the theory of laterization and calling attention to the laterites of India, South America, and Africa, refers to the alteration of serpentine in words which to all intents and purposes describe this earthy variety of the ore.

He says:

> "The alteration of the eruptive rocks, formed of ferromagnesian silicates, also produces ferruginous laterites similar to those described above, of a deep brick red color. In the notably uniform composition of this red earthy mass so characteristic of tropical regions, the variations it presents depend solely on the greater or less proportion of the hydrated oxide of iron that it contains."

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[5] *Grande Encyclopédie Française.*

2. The ore also occurs in small shot-like particles, generally concentrated on the surface by the action of water, but frequently imbedded in a matrix of the earthy ore to the depth of 8 or 10 ft., and occasionally even deeper. At Moa these pellets or nodules are frequently coarser than at Mayari and such coarser nodules are found over the entire surface of considerable areas.

3. A third form is the result of the cementing together of the smaller nodules and "shot" ore into boulders and masses. On the Mayari plateau in several localities, at the sources of streams where there is an excess of water with little velocity of flow, the ore has been cemented by the action of the sun, air, and water (with dissolved and re-precipitated iron oxide as a cementing material), to form beds or layers of so called hard ore. These cover many acres in different portions of the Mayari plateau; but are rarely more than 2 or 3 ft. thick, and do not differ in analysis from the remainder of the ore-body. I have not observed similar beds at Moa, but hard ore has frequently been encountered in the borings, which may indicate a continuous layer or may prove to be only isolated boulders.

Perhaps the most striking characteristics of the ore, and certainly of paramount importance commercially, are the great extent of territory that it covers and the entire absence of over-burden.

The ore-beds are continuous for miles, and in Moa alone there are more than 70 sq. miles of a nearly-continuous ore-body, broken only by patches of low-grade ore and barren areas, where rivers and streams have carried away the ore and exposed the underlying serpentine.

The Moa deposit lies on the northern slope of the range and extends from the water's edge inland for a distance of nearly 10 miles in an almost unbroken stretch. Here it is more or less broken by the topography of the mountain formation, but beyond the first range and an intervening valley it is again found on a plateau 15 miles inland. Along the coast there are occurrences of ore all the way from Moa nearly to Baracoa. Here are the Taco and Navas fields, to which reference has been made. What is known as the Moa district extends nearly 10 miles east and west. The mountain spurs come down in some instances to the sea, and the ground is therefore cut by

drainage into a series of ridges and valleys. There are, however, great stretches of gently-rolling country.

The absence of over-burden other than the growth of timber and underbrush is an important characteristic. The pine timber usually has from 6 to 8 ft. of tap-root and seems unable to penetrate the ore further, as the root "brooms" at this point. This would appear to account for the fact that trees after about 25 years of growth do not increase in size and are rarely found as large as 2 ft. in diameter. The forests of other woods have shallow roots spreading over the surface of the ground. There is not enough vegetable humus to make any significant difference in the analysis of the surface-ore as compared with the deeper ore. It may be mined literally from the grass-roots down.

Discovery.

Both in the Mayari and Moa districts the so-called "King's Highway," as the rough pony-trail is somewhat pompously termed, runs directly over the ore. In the Mayari field the Spanish troops during the 10 years' war built a small fort, complete with its moat and bastions, entirely of the iron-ore. This was intended to guard the telegraph-lines, which followed the trail, as well as the trail itself. It is evident, therefore, that in both districts, as well as in Taco and Navas, the red earth was well known, but had never been recognized as ore. The discovery that it was ore was a gradual process. More than 20 years ago a number of claims were located, or in the Spanish term "denounced," in the Moa district. They were near the coast and were confined chiefly to the occurrence of boulders and hard ore, and as the quantity of these was not great, and as some of the earlier samples were extraordinarily high in chromium, the district did not attract any attention and some of the claims were abandoned. Again, about 8 years ago, interest was renewed in the deposit and a number of claims were denounced. To my knowledge, at least three different engineers reported on the deposit at different times and their conclusions were unfavorable.

This result seems to have been due to the fact that everybody considered only the shot ore and boulders. The possibility of concentrating the former by washing it from its earthy

matrix was discussed more than once. It had not yet dawned on any one that the earthy matrix itself, that great mantle covering so vast an area for a depth of 18 ft. and more, was really just as good quality ore as the concretionary forms on which attention has been centered. The report of a Spanish engineer that there were 35,000,000 tons of ore on the claim which he examined at Moa, was received with amused tolerance. He was, in fact, not far from the truth, though he himself did not claim that the earthy mass was ore, and incorrectly relied on the shot ore and boulders to yield his tonnage.

It was in the Mayari district, where systematic explorations were undertaken as early as 1904, that the truth came to light. Even here the earliest explorations were conducted under my direction on the hard-ore beds or layers already noted. A casual sample in a sink-hole directed attention to the quality of the red earth as being in no way different from that of the *planchas*, as the natives term the beds of "hard" ore. Even after the quality of the red earth was known, the yellow ore lying below it looked so like a simple clay that it was neglected, until systematic sampling and analysis developed the fact that it was as high, and probably higher, in iron-content than the hard ore and the red earth.

Thus it slowly became apparent that, from the surface to the underlying serpentine, the whole mass was in effect homogeneous, the occurrence of concretionary forms being due to local conditions that did not alter in any marked degree its chemical composition, and the variations in color being due to its more or less hydrated condition. Then, for the first time, the enormous extent and vast commercial importance of these ore-bodies were grasped.

Following this discovery in the Mayari district, a great number of claims were denounced in Moa, but no explorations were undertaken until the Spanish-American Iron Co., which owned and had explored the Mayari district, began the exploration of its claims at Moa.

Explorations.

This work began in December, 1905, and was completed in July, 1906. During a part of that time two full parties were

in the field, more than 50,000 acres of ore-land was examined, the district was mapped, hydrographic surveys were made of Moa bay, and several thousand samples of ore were taken and analyzed.

The nature of the ore-body lends itself to exploration by a method which was developed at Mayari, and which merits description here.[6] The ore can be bored with an ordinary carpenter's auger, and its consistency is such that the entire shaving remains in the auger, which is withdrawn and cleaned every four complete turns, and thus an accurate sample of the ground bored is secured. In spite, or better, because, of the high percentage of water in the ore the hole does not cave, and by cleaning it frequently, to avoid getting scrapings from the upper portions in the lower samples, the correctness of these can be assured. By using sectional rods, with a screw-thread at each end and connected by a sleeve-nut, holes have been bored exceeding 80 ft. deep.

The men acquire great facility in handling the augers and can bore very rapidly, covering a wide area each day. In no other way would it have been possible to explore this large territory except with much difficulty and at great expense. Occasional pits are valuable as a check on the borings and to permit a study of the ore at various depths, but no system of pits or tunnels could have done the work so thoroughly in the comparatively short time that the borings accomplished these results.

The boring system is dwelt on here as, when actual mining begins, it not only affords an inexpensive manner of determining in advance the exact quality of the output for any given length of time, but also, from the knowledge thus derived of the topography of the underlying rock, it enables the mining-work to be planned for years in advance.

In this manner more than 900 borings were made, and samples were taken of every 6 ft. of each boring, and sometimes oftener. In a number of cases these borings were stopped by hard ore. A means of penetrating this has since been devised, and, in practically all cases, the usual ore was found below the shell of hard ore, frequently better in quality than that found above.

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[6] This volume, p. 146.

The borings were spaced generally at intervals of 1,000 ft., although in some cases rows of borings were separated from one another by a greater distance. The claims thus systematically explored were the Sagua, Baracoa No. 2, Yamaniguey, Moa, Lirio, Cabañas, Punta Gorda, and Yagrumaje, which, for convenience, are referred to as the "Moa group," and which cover a total area of 13,832 hectares, or 34,179 acres. A few borings were made on most of the other claims then existing in the district; this work covering another 8,572 hectares, or 21,181 acres. In 1910 systematic borings spaced 250 m. apart were made over the claims, Juan Manuel, Frasco, Guarico, Guarico Primero, Guarico Segundo, Ysabel, Ysabel Primero, Ysabel Segundo, Esperanza, and Esperanza Primero, which form a compact group just south of the great Punta Gorda claim, covering 5,460 hectares, or 13,492 acres, and known as the "Rodrigo group."

The ore-area explored on the Moa group was 8,100 hectares out of a total of 13,832. Nearly 1,000 hectares of additional ore-ground were not explored in 1906, as they were in the southerly portion of the Punta Gorda claim, and were separated from the original explorations by barren ground. They were subsequently explored in connection with the group of claims just south of the Punta Gorda.

Over the area of 8,100 hectares or, more exactly, 874,000,000 sq. ft., the average depth of the ore was found to be 18.1 ft. As the ore varies in density, a number of tests were made to determine the number of cubic feet to the ton. This varied from 15 to 24, with an average of 18.5; but, for the purpose of calculating tonnage, 20 cu. ft. of the ore in place has been taken as representing a ton.

The 874,000,000 sq. ft. of ore 18.1 ft. deep, at 20 cu. ft. to the ton, would give 791,000,000 tons, but as a matter of fact the borings near the edge of the ore-body are shallower and represent smaller areas, and, in calculating the ore-tonnage, the depth of each boring was multiplied by the area it represented and the product divided by 20, which yielded a total tonnage of 803,000,000 tons in the Moa group.

It might be argued that the great tonnage above shown is based on too small a number of borings; and such a point of view would not be unnatural to an engineer accustomed to

work on veins or limited areas; but a simple inspection of the ground by an engineer, having before him that proof of the homogeneity of the ore-body which chemical analysis affords, will convince him of the accuracy of the method. Furthermore, this has been demonstrated in actual practice. At Mayari the earlier borings were made at intervals of 100 ft., but the ore proved so homogeneous, in character and analysis, that this distance was increased to 300, 500, and finally to 1,000 ft. The results thus secured were subsequently checked and confirmed by borings, 25 and 50 ft. apart, on four limited areas, widely separated from one another, and each representing several million tons. Furthermore, in the Moa deposit, borings made in 1908, every 100 m., over two claims representing 340 hectares, have proved up 45,000,000 tons. In the explorations I made in 1906, these two claims were included in the number in which only a small amount of work was done, 18 borings being made on the two claims in question. The thorough explorations referred to check the 18 borings very closely as to depth and show a better quality of ore than was found originally.

As a final proof, the entire exploration-work, by which the 803,000,000 tons were developed, was checked in 1910 by an independent engineer, Dwight E. Woodbridge, who rebored about one-half of the original work, and also made intermediate borings, which confirmed the results from surrounding borings. He also explored, for the first time, nearly 1,000 hectares on the Punta Gorda claim, and about 540 hectares on smaller claims not previously explored, checking very closely the original results and increasing the total tonnage to 865,000,000 tons.

In the Rodrigo group, which lies to the south of the Punta Gorda claim, complete explorations have developed 224,000,000 tons of ore.

The estimated ore in the mining-claims which were less thoroughly explored, but where the results may be regarded as fairly accurate, gives a total of 280,000,000 tons. We have thus in the Moa district proper as the result of my explorations 1,307,000,000 tons of ore, which total has been increased by the work of others. These are startling figures, but their accuracy is unhesitatingly affirmed.

Such a body of ore lying on the surface, with no further

over-burden than the pine timber and hardwood forests, extending from the shore of a deep-water bay over a country perfectly accessible for a railroad, apart from its interest to the scientist, compels the attention of the engineer for its commercial possibilities.

Quality of the Ore.

The iron is present in the lower levels practically all in the form of limonite and possibly other hydrated oxides. Near the surface it has parted with some of its combined water and oxygen, and hematite and magnetite predominate over the limonite. Alumina and chromium are fairly constant, but nickel and cobalt increase materially in the lower levels.

An uncompensated average of samples over the entire area, including all grades of material, gives the following results:

	Per Cent
811 samples, Fe	41.32
568 samples, SiO_2	7.22
568 samples, Al_2O_3	14.81
568 samples, Cr	1.58
202 samples, P	0.012

The nickel and cobalt vary from 0.44 to 1.28, with an average of 0.8 per cent.

The natural basis of comparison for this ore is with that of the Mayari deposit, and it is found that the former is rather more homogeneous. While in Moa the amount of ore carrying less than 30 per cent. of iron (and consequently of a quality that would be discarded in operating the mine) is about the same as in Mayari, there is a considerably greater proportion of Moa ore between 30 and 40 per cent., as shown in Table I., prepared in 1906:

Table I.—*Comparison of Mayari and Moa Ores.*

Composition Analysis.	Percentage of all Samples. Mayari Per Cent.	Moa. Per Cent.
From 10 to 20 per cent. of iron,	4	1
From 20 to 30 per cent. of iron,	2	4
From 30 to 40 per cent. of iron,	6	27
From 40 to 43 per cent. of iron,	6	20
More than 43 per cent. of iron,	82	48

The figures given in Table I. include all of the ore-body. Tables II. and III. show the quantities of this ore that can be regarded as commercially available.

TABLE II.—*Commercial Ores of Moa Group.*

	No. of Borings.	Average Depth Feet.	Iron-Content (Compensated for Tonnage). Per Cent.	Tons.
Less than 30 per cent. of iron, .	24	17.9	23.3	34,321,000
From 30 to 35 per cent. of iron,	35	17.4	32.2	40,670,500
From 35 to 40 per cent. of iron,	85	16.5	37.7	92,501,500
From 40 to 45 per cent. of iron,	182	18.7	42.6	311,379,750
45 per cent and upward, . .	203	18.7	46.3	305,580,750
Borings not analyzed, but assumed at average analysis of other borings, ...	19	14.7	42.0	18,657,500
Total, ...	548	18.1	42.0	803,411,000
Omitting all containing less than 30 per cent. of iron, . . .	524	......	42.9	768,790,000
Omitting all containing less than 35 per cent of iron, . . .	489	......	43.5	728,119,500
Omitting all containing less than 40 per cent. of iron, . . .	404	......	44.3	635,618,000

TABLE III.—*Commercial Ores of Rodrigo Group.*

	No of Borings.	Average Depth. Feet.	Iron-Content (Compensated for Tonnage). Per Cent.	Tons.
Less than 30 per cent. of iron, .	14	23.6	22.9	9,122,600
From 30 to 35 per cent. of iron,	37	18 4	33 0	21,961,600
From 35 to 40 per cent of iron,	71	21.4	37.3	49,496,200
From 40 to 45 per cent. of iron,	142	21.3	42.8	96,791,700
More than 45 per cent. of iron,	68	20.4	46.7	46,821,700
Total, ...	332	20.8	40.6	224,193,800
Omitting all containing less than 30 per cent. of iron, . . .	318	20.8	41.4	215,071,200
Omitting all containing less than 35 per cent. of iron, . . .	281	21.1	42.3	193,109,600
Omitting all containing less than 40 per cent. of iron, . . .	210	21.0	44.0	143,613,400

Attention is called to the fact that all the determinations of Tables I., II., and III. are of the ore dried at 212° F., and still containing its combined water. Furthermore, they represent only the soluble iron, as, on account of the chromium present, the determination of total iron required fusion and was slow and inconvenient, especially when the laboratory-work was done in the field, as was the case with some of the explorations.

To the figures given, from 0.5 to 1 per cent. should be added for the insoluble iron, and 0.8 for nickel, a total of 1.5 in all.

Taking from Tables I., II., and III. all the ore containing more than 30 per cent. of iron, we have:

	Tons	Per Cent.
Moa group, ...	768,790,000	42.9
Rodrigo group, ...	215,071,200	41.4
Compensated, . . .	983,861,200	42.57

Add to this content of soluble iron 1.5 for insoluble iron and nickel and we have 44.07 per cent. The combined water exceeds 14 per cent., but 12 per cent. has been assumed as more conservative. Expelling this water, we have 44.07 divided by 0.88, or 50 per cent. of metallic units in this 983,000,000 tons of ore.

The 280,000,000 tons, estimated in other claims less accurately explored, are a part of the same great ore-body, continuous and homogeneous with the remainder and separated from it by arbitrary survey-lines only. It is, therefore, reasonable to assume that its average quality is substantially the same.

Assuming this proportion, we have 268,000,000 tons for these claims, a total of 1,251,000,000 tons of ore with 50 per cent. of metallic units of iron and nickel.

It must, however, be borne in mind that the above tonnage represents the weight of the ore as it lies in the ground; while the analysis represents ore from which all moisture, both hygroscopic and combined, has been expelled.

In addition to from 10 to 14 per cent. of combined moisture, the ore contains from 25 to 30 per cent. of hygroscopic moisture, a total of from 40 to 42 per cent. of water in the average ore. There is nothing in the appearance of the ore to indicate this very high percentage of moisture. Shafts and tunnels stand without timbering, and after several years still show the marks of the picks. In open-cuts, where the ore is in vertical faces, it drops off after exposure to the sun and rain.

The tonnage of ore actually containing, when dried, 50 per cent. or more of metallic units of iron and nickel, as calculated above, must, therefore, be taken at 60 per cent. of the total tonnage of ore in the ground in its natural state, or 750,000,000 tons. This tonnage refers only to the ore developed by the explorations described above. There are some claims near the sea

which I have not explored and the ore 15 miles inland I have not even visited. No attempt is made to estimate the additional tonnage these represent.

Commercial Availability.

Since there must be always two sides to the shield, it is not surprising that an ore, with so many features favorable to its commercial exploitation, should also present problems that involve extensive and painstaking experiment for their solution. Its clay-like consistency made it uncertain how it would act in the steam-shovel dippers or excavator-buckets, difficult to remove from cars by dumping, and practically impossible to stow in bins or any other device involving its removal from the bottom of the pile.

The high percentage of moisture, and the consequent transportation-charges and duties on 40 per cent. of water, necessitated drying the ore, and the extreme fineness of most of the dried ore made it necessary to carry the process further, and, by increased temperature, produce incipient fusion and convert the ore into nodules suitable for the blast-furnace.

In the furnace the high alumina-content complicated the slag-calculations, and once pig-iron was produced, the elimination of the chromium was essential to the production of satisfactory steel.

A detailed description of how these several problems have been met would form a paper of some length. It is sufficient here to say that the Pennsylvania Steel Co., the parent company of the Spanish-American Iron Co., has worked out practical solutions of all these difficulties in the case of the Mayari ores, and the results are applicable to the similar Moa ores.

At Mayari, the ore is handled by shovels, or by scraper-excavators, and does not cling to the dipper or bucket. Special railroad-cars capable of being tipped to an angle of 90° are used to transport the ore, which falls from the car, leaving the inner surface nearly clean. When the ore must be handled subsequently, this is done by grab-buckets from gantries or moving bridges.

Kilns, resembling the ordinary cement-kiln, are used to produce nodules, which have proved highly satisfactory for use in

the blast-furnace; and it has been demonstrated that a certain amount of undried ore can be mixed with the nodules in the furnace-charge without producing any abnormal amount of flue-dust. In spite of the high alumina, the blast-furnace slag gives no trouble.

The partial or complete elimination of the chromium and the production of a satisfactory steel were accomplished after patient experiment. Steel rails made from this ore have demonstrated their superiority over ordinary rails, by actual use on the Horseshoe Curve of the Pennsylvania railroad. For more than a year the Pennsylvania Steel Co. and the Maryland Steel Co. have manufactured commercially, from Mayari ore, a steel which, by reason of its nickel-content and low phosphorus, is superior to the ordinary Bessemer and open-hearth products.

Harbor.

A safe and sufficiently large harbor is indispensable to successful exploitation. The so-called bay of Moa is apparently directly exposed to the prevailing NE. trade-winds, but is in reality protected by a line of coral reefs which form an effective breakwater and render the harbor a safe anchorage for the largest vessels.

There is a wide entrance between the reefs, and soundings have shown the existence of deep water to within 1,000 ft. of the shore.

Cost of Plant.

The deposit offers so many attractive conditions that some figures on the cost of opening it should prove of interest.

In the absence of accurate surveys and of borings to determine the nature of the bottom of Moa bay, only an approximate preliminary estimate can be presented.

The first step in opening the mines should be the purchase of a steam-lighter of from 150 to 200 tons capacity, and capable of going to sea in all ordinary weathers, with a speed of from 7 to 8 miles an hour. Such a vessel would be indispensable in preliminary operations, carrying supplies and men from Nipe or Baracoa, for surveys, establishing camps and installing the first machinery and a wharf to deep water. When this

has been done, it should be arranged to have Moa made a port-of-entry, so that freight-steamers could bring machinery and supplies directly to Moa bay. A steam-lighter is suggested as being safer than an ordinary tug with lighters. The latter are used under similar conditions on the south coast of Cuba, but the sea is generally rougher on the north coast where the trade-winds prevail.

A telephone-line should be built to Mayari, near Nipe bay. This will probably be done by the government when permanent construction is begun at Moa.

Further requirements are, an ore-dock or other loading-device, apparatus for the discharge of coal, a short railroad, shops, power-plant, nodulizing-kilns, water-supply, telephone-lines, ice-plant, and the necessary buildings to house the employees and laborers.

Although I have prepared figures on the necessary installation in detail, the estimate is here given as a total, because, while reasonably close in the aggregate, there are not available sufficient data for accuracy in the details of the installation. It is estimated that the cost of opening the mines and equipping for a production of from 45,000 to 50,000 tons of nodules (requiring say 75,000 tons of crude ore) per month, and from 25,000 to 30,000 additional tonnage of crude ore, would be between $3,000,000 and $3,500,000.

After the work of construction is completed the amount of unskilled labor employed will be comparatively small, as the mining will be done by shovels and excavators. The mining and railroad laborers employed in Cuba are almost exclusively Spaniards, and these generally *Gallegos*, or natives of the Province of Galicia. Locomotive engineers, machinists, carpenters, etc., are usually Cubans. The Spanish laborers are sober, industrious, and generally easily handled. Strikes are of rare occurrence, and labor unions do not exist. The ruling rate of wages is $1 per day; but contract- and task-work are the rule, so that the men can earn from $1.30 to $1.50 per day of 10 hours.

Timber for railroad-trestles, ties, and general construction is available on the ground. The pine timber, while not equal to long-leaf yellow pine, is perfectly good for all ordinary building purposes.

It is believed that nowhere else in the world is there a continuous iron-ore deposit of such magnitude. The size of the ore-body gives a character of permanence unusual in mining-operations, the nature of the ore insures a minimum mining-cost, the proximity to tide-water affords unusual transportation-facilities, and the presence of nickel in the ore adds greatly to the value of the finished product.

In these days, when the ownership of a supply of raw material is recognized as indispensable to the successful operation of any great steel-works, the field offers an absolutely unique opportunity to a plant on or near the Atlantic coast.

Origin of the Iron-Ores of Central and Northeastern Cuba.

BY C K. LEITH AND W. J. MEAD, MADISON, WIS.

(Wilkes-Barre Meeting, June, 1911)

ONE of the most significant developments in the iron industry in recent years has been the discovery and opening of enormous reserves of low-grade ore in eastern and northeastern Cuba. The two principal fields are the Mayari and the Moa, situated on Nipe bay, in the Province of Oriente. A less well-known district of the same type is that of Baracoa, at the east of the island, and another is in Camaguey Province in central Cuba. In the comprehensive estimates of the iron-ore reserves of the world, published during the summer of 1910 by the International Geological Congress in Sweden,[1] these Cuban deposits are estimated at about 2,000,000,000 tons. Certain it is that the reserve is a large one. The Spanish-American Iron Co., the Juragua Iron Co. (Bethlehem Steel Co.), the U. S. Steel Corporation, and others have been active in this exploration. The Spanish-American Iron Co. has established a port for the handling of these ores at Felton, has built a railway 16 miles to the ore-fields, and has opened up deposits for steam-shovel mining. Its expenditures have reached an aggregate of about $6,000,000 in preparation for the handling of these ores. Shipment of the ore to Sparrow's Point, Md., has begun and may be expected to rise to a considerable amount in the near future.

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[1] *The Iron Ore Resources of the World, an inquiry made upon the initiative of the Executive Committee of the Eleventh International Geological Congress,* Stockholm, vol. ii., p. 795 (1910).

The ore reaches this point at a cost per unit of iron considerably lower than Lake Superior ores, and the presence of a low percentage of nickel gives a desirable steel. It is not the purpose of this article to consider these ores from a commercial stand-point, but rather to discuss their origin. If our conclusions are correct, they throw light on the character of the deposits which should have some commercial significance.

I. The Moa and Mayari Deposits.

Much of the interior of Cuba is a plateau from 1,500 to 2,000 ft. above sea-level, and locally higher or lower, on which the ores rest. On the steep slopes descending to the ocean and in drainage-channels in the interior the ores are thin or altogether lacking. The deposits as a whole constitute a nearly horizontal mantle, from a few inches to 80 ft. thick, over the surface of the serpentine country-rock. The lower contact is irregular. Erosion has exposed the country-rock in the valleys and slopes. The ores are dominantly limonite, containing more or less hematite, magnetite, and intermediate hydrates of iron near the surface. Metallic iron averages about 46 per cent. While locally variable, there is a general tendency for the iron to decrease slightly towards the surface, and to maintain its average grade, or better it, towards the bottom. The content of free water averages from 25 to 30 per cent., the combined water from 10 to 15 per cent. When dried and dehydrated in a nodulizing-plant at Felton (similar to a cement-kiln), the moisture and combined water are driven off sufficiently to bring the content of metallic iron up to between 50 and 55 per cent. Phosphorus is for the most part below the Bessemer limit. Nickel and cobalt are present in quantities ranging up to 1.5 per cent. throughout the deposits, especially in the middle part. The presence of these metals is advantageous to the quality of the steel produced from the ore. Chromium ranges up to 2.5 per cent., the higher amounts being reached towards the middle and lower parts of the deposits. Its economical elimination in smelting has been demonstrated.

The principal impurity is bauxite (and gibbsite), which, however, gives no trouble in smelting. In smaller amount is kaolin. Bauxite increases in percentage towards the surface, while kaolin decreases. The percentage of kaolin is not higher than in certain Mesabi ores.

The lower parts of the deposits are soft and earthy in texture and of a yellow color, grading up to a dark red in the upper parts, which are granular, consisting of small pellets ranging locally up to 0.5 in. or more in diameter, in a matrix of soft iron oxide and bauxite. Directly at the surface, in places where the ore has not been disturbed by erosion, the granular ore has been cemented or case-hardened by infiltration of iron salts into *planchas*, or sheet-deposits, up to 4 ft. thick. Bedding is nowhere to be seen in the ore.

1. *Source of Mayari and Moa Ores.*

Geologists are substantially agreed that the ores of the Moa and Mayari fields are residual or mantle-deposits resulting from surface-alterations, in place, of serpentine rock, which in turn probably represents the alteration of some other rock like a peridotite not yet disclosed by underground explorations. With the serpentine there are present very minor quantities of intrusive dike-rocks high in alumina, which by the surface-alterations yield clay, not iron-ore. Iron-ore of the Mayari type is found not only in Mayari, but in other parts of Cuba where serpentine is known. In addition to practical identity in distribution of the iron-ore and serpentine, the iron-ore shows mineralogical, chemical, and textural characteristics which are the normal result of alteration of the serpentine rock at the surface.

Notwithstanding the general consensus of opinion as to the origin of the Cuban ores of this type, the question was recently raised as to the proper classification of the ores under the Cuban laws, and it became desirable to establish more definitely, so far as possible on a quantitative basis, the derivation of the iron-ore deposits from the serpentine by weathering *in situ*. The results of this work are presented below.

Thousands of analyses made by the Spanish-American Iron Co. have been available, and, in addition, 20 analyses have been especially made under the supervision of Mr. Mead from samples personally collected.

The changes from serpentine to ore may be considered (*a*) in terms of volume of minerals and rock, and (*b*) in terms of weight.

a. Consideration of Alterations from Serpentine to Ore in Terms of Volume.—In Fig. 1 the gradation in composition of the ore from the surface downward, and the changes in composition of the serpentine rock itself during its alteration to ore, are shown graphically. The figures platted are from actual analyses and measurements of pore-space, and represent definite, indisputable facts in a typical and average case. The minerals

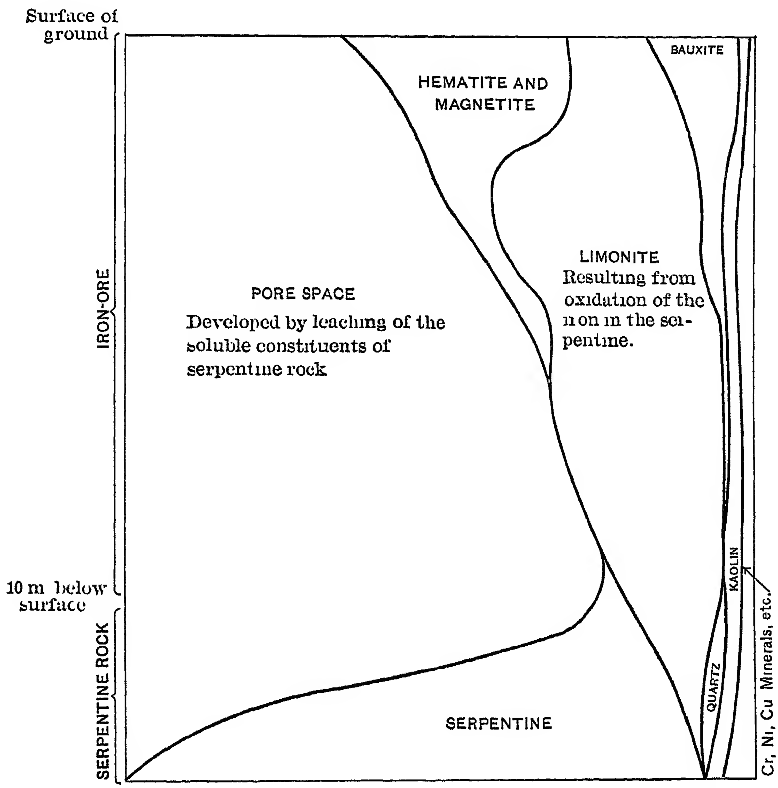

FIG. 1.—DIAGRAM SHOWING IN TERMS OF VOLUME THE VARIATIONS IN MINERAL COMPOSITION AND PORE-SPACE IN A TYPICAL MAYARI ORE-BODY, FROM THE SURFACE DOWNWARD INTO THE SERPENTINE ROCK.

Based on a series of 11 complete physical and chemical determinations on samples taken at approximately equal vertical intervals.

have been calculated from the chemical analyses and reduced to terms of volume, to permit the consideration of pore-space. The iron has been calculated as the minerals hematite and limonite, though this does not necessarily preclude the existence of

hydrates intermediate between hematite and limonite, or possibly higher hydrates than limonite. Hematite includes magnetite. The diagram is based on determinations made on samples of material taken at approximately uniform intervals from the serpentine rock upward through the ore to the surface. To avoid misinterpretation of the diagram, it may be noted that in the rock and ore the several minerals and pore-spaces are, of course, intricately mixed, not separate as in the diagram. The serpentine rock (below) consists dominately of the mineral serpentine, which is hydrous silicate of magnesia and iron. Kaolin, and chromium-, nickel-, and cobalt-minerals are subordinate constituents. In approaching the ore, pore-space develops because of leaching of silica and magnesia from the serpentine; and limonite appears because of oxidation of the iron of the serpentine. A small amount of quartz appears because the silica derived from the breaking-down of the serpentine is not all immediately carried away. Coming to the ore nearest the serpentine, it appears that the serpentine has been entirely destroyed, and the proportion of limonite and pore-space increased. Quartz is entirely lost. Following the ore towards the surface, the most conspicuous change is seen to be the lessening of pore-space, thereby increasing the relative volumes of the other constituents. In the middle of the deposit hematite (and magnetite) begins to appear with the limonite, due to dehydration and deoxidation of the limonite, and these minerals increase gradually to the top of the deposit, where they are more than twice as abundant as limonite. Kaolin towards the surface gives way to bauxite, due to the loss of the silica from the kaolin. Bauxite increases towards the surface. Nickel- and chromium-minerals persist through all the alterations of rock and ore, affording easily recognizable evidence of derivation of ore from the rock.

b. Consideration of Alterations of Serpentine to Ore in Terms of Weight.—The foregoing discussion of alteration is based on volumes of minerals and pore-space. If we consider the alterations in terms of weight of constituents, rather than volume, by graphic methods, further significant facts appear.

The alumina has apparently remained constant, as would be expected of this most-insoluble constituent. If lost at all, it has been so much less so than the other substances that it may

serve as a standard against which loss of other constituents may be measured. On this basis, it appears that iron, during the alteration of the rock to the ore, and in the lower part of the ore-bodies, has been lost as little as alumina; in other words, it maintains its proportions with the alumina. Towards the top of the ore-body it has been lost relative to the alumina, thus increasing the per cent. weight of alumina in the mass. In the middle portions of the ore-body iron has actually increased in proportion to the alumina, due probably to redeposition of iron dissolved near the surface.

Silica is continuously lost throughout the operation, both from the breaking-down of the serpentine and from the kaolin which alters to bauxite. Kaolin ordinarily holds its silica very firmly. Its loss from kaolin is a well-known peculiarity of alterations in tropical climates, known as lateritic alterations. Absence of any free quartz in the ore distinguishes this deposit from many bog ores. Magnesia, readily soluble under surface-alterations, has been completely lost and is not found in the ore. The combined loss of the silica and the magnesia substantially account for the increased proportion of the iron and alumina.

In a typical case 100 lb. of serpentine rock contains approximately 1.5 lb. of alumina and 10 lb. of ferrous oxide. When the magnesia and silica are removed in solution and the iron oxidized, there remain approximately 11.75 lb. of limonite, 3.8 lb. of bauxite and kaolin, and, at the most, 2 lb. of minor constituents. This residual of 17.55 lb. contains 7.8 lb., or 44.4 per cent., of metallic iron, and is an iron-ore.

2. *Textures of the Ores and Serpentine.*

In the change from serpentine to ore above described, large pore-space is developed in the ore, due to removal of material in solution. This pore-space may or may not be filled with free water. The large amount of this pore-space, in some ores as high as 80 per cent., is to be explained by the fact that the leaching of substances molecularly combined with the iron and alumina leaves extremely minute and irregular, though numerous, pores of such shape and dimensions as to enable the ore to stand under its own load.

Towards the surface pore-space in the ore lessens, due to

cementation of the ore and the slump accompanying the mineral alterations of this zone.

The iron near the surface also congregates into granules, ranging up to 0.5 in. or more in diameter. These seldom extend more than 10 or 12 ft. from the surface, beneath which the texture is fine-grained and soft. Where the surface has been undisturbed by erosion, and therefore has been subjected to chemical changes for a long time, these granules are likely to be large, and they are likely also to be cemented by iron oxide into sheet-like deposits from a few inches to several feet in thickness, locally known as *planchas*. The zone of granules is also the zone in which hematite and magnetite are developed, and the zone containing evidence, both in the nature of the cements and in the analyses, of the solution of iron, chromium, nickel, and silica. These facts would seem to correlate the formation of these granules with the solution going on at the surface. It is to be noted that the granules extend to about the depth to which the vegetation and roots extend, also that around the roots iron has been locally dissolved and reprecipitated as a casing to the roots; many of the continuous tube-like holes in the *plancha* ores are not improbably root-holes. A reasonable inference from these facts, though not an established conclusion, is that the decay of vegetation yields organic acids (solvents for iron salts) which have been a prime cause in the local solution and transfer of the iron. The granular ores seem to follow even minor irregularities of the surface developed since the ore was formed, indicating the comparative recency or contemporaneity of the process of the formation of the granules.

3. *General Consideration of Alterations.*

The fact is to be emphasized that the above-described process of alteration affects all rocks at the earth's surface, but with varying results, depending on the original ingredients of the rock and other factors, and that ores in general have their principal values developed through these processes. Location at the surface is essential to the change. It is a breaking-down process technically known as katamorphism, a process resulting in simplification of mineral composition, elimination of part of the constituents through solution, and mechanical disintegration. It is contrasted with a process which goes on far

below the surface, anamorphism, resulting in the development of complex from simple mineral compounds and a more compact physical structure. When a rock is brought to or near the surface it is subjected to the chemical and mechanical action of air and water. Most minerals of rocks are more or less soluble in surface-waters, the more-soluble portions being carried away and the less-soluble substances remaining. The least-soluble constituents are ferric oxide and alumina-minerals, such as bauxite and kaolin (clay). Silica or sand is also difficultly soluble, but is more soluble than iron oxide or kaolin. The average igneous rock of the earth contains about 15 per cent. of alumina (the distinctive constituent of clay) and 3 or 4 per cent. of iron, and hence the common result of rock-decay at the surface is a clayey soil carrying some iron oxide. The serpentine rock, however, contains a remarkably small amount of alumina, in a typical case 1.5 per cent., and about 10 per cent. of ferrous oxide. Therefore, when the soluble constituents, magnesia and silica, are removed in solution and the iron oxidized to ferric oxide, the result is a porous mass of iron oxide containing minor amounts of bauxite and kaolin, together with small amounts of nickel-, cobalt-, and chromium-minerals; in other words, is iron-ore rather than a soil.

4. *Contact of Ore and Serpentine.*

Explorations and mining have seemed to disclose a fairly-narrow zone of transition between the serpentine and the ore. Yet the evidence of gradation is indisputable. Analyses show gradations on both sides of the contact, towards the ore in the rock, and towards the rock in the ore. The break at the contact is less largely in the texture than in the composition, but even in the texture there is evidence of gradation. Residual kernels of the unaltered rock are frequently found in the ore. Finally, the irregular nature of the contact, the presence of basin-like depressions, and the absence of regular systems of erosive channels, are characteristic of residual surfaces from weathering *in situ*. It is really an etched surface of solution.

A fairly-sharp contact of rock and ore is nowise exceptional in residual deposits resting upon their parent rocks. They may be noted where residual clay rests upon a granite. More significant, perhaps, is the parallel in the Lake Superior region,

where the ore usually rests in sharp contact upon the ferruginous cherts and jasper, from which the ores are derived by elimination of silica.

5. *Comparison of Mayari and Mesabi Iron-Ores.*

Notwithstanding many differences in form, geological relations, mineralogical and chemical composition between the Mayari deposits and the Mesabi deposits, they have many essential features in common. In both, limonite, hematite, and magnetite are present, and in roughly the same proportions. Crystalline and earthy varieties are in similar proportions. Average content of clay is almost identical. The Mayari deposits contain bauxite in addition; the Mesabi deposits contain quartz. They are both substantially residual products, in place, of the alteration of the rocks upon which they rest, though their original rocks are of quite different nature. Both were necessarily developed at the rock-surface. They are both the results of katamorphic processes affecting all surface-rocks. In both cases there has been a survival at the surface of insoluble substances fittest to withstand a surface-alteration. Both owe their economic importance to the fact that iron oxide happens to be the substance most permanent under surface-conditions, and has therefore accumulated in large deposits at the rock-surface through the elimination of other constituents which were with them when their alteration process started. In both, development of the ore-deposits has been accompanied by destruction of original textures, increase of pore-spaces, minor solution and redeposition of the iron, and local cementation through this means. In both, the ores are at the surface, and the greater dimensions of the deposits are horizontal, making it advantageous to mine by a steam-shovel.

The ores of the two districts differ in that those of the Cuban district have undergone a single direct concentration from an igneous rock, while those of the Mesabi district are the result of similar concentration, in place, of a peculiar type of sediment, high in iron, known as an iron formation, which in turn was ultimately derived from an igneous rock and has been transported to its present position.

II. The Camaguey District.

In the Camaguey district deposits of iron-ore of commercial grade and quantity are well exposed at the surface and in

shallow test-pits. They have been penetrated also by hand-drilling, but detailed results of this work, with analyses, were only partly available to us at the time of our visit. Like the Moa and Mayari ores, they are mantle-deposits of large surface-area, with a thickness ranging up to 15 ft. or more, resting on the surface of the serpentine country-rock. Erosion has removed the ore and exposed the country-rock in the valleys and on the slopes. A large part of the ore averages more than 40 per cent. of iron. Downward the grade becomes poorer. In the surface-samples limonite and hematite (with magnetite) are about in equal quantity. From the surface downward the proportion of limonite to hematite increases. Phosphorus is for the most part above the Bessemer limit. The surface-ores contain about 0.5 per cent. of nickel and between 1 and 2 per cent. of chromium. The economical elimination of the chromium in smelting has been demonstrated in the similar Mayari ores.

The principal impurities are bauxite (and gibbsite), kaolin, and free chert. Bauxite increases in importance towards the surface, kaolin and chert decrease. In abundance of chert fragments these ores contrast with the Mayari and Moa ores.

The deposits in general consist of irregular fragments and pellets of iron oxide in an earthy matrix. In the lower parts of the deposits the fragments may reach a diameter of several inches up to a foot. Towards the surface they tend to decrease in size, and in the zone of the grass-roots the ore is generally of a fine, earthy texture. Locally, however, in this upper zone there is a development of granules. Directly at the surface in places where the ore has not been disturbed by erosion, the granular ore has been locally cemented by infiltration of iron salts into *planchas*, or sheet-deposits. Bedding is everywhere absent in the ore.

Towards the surface the ores have undergone some secondary alterations, as follows:

a. The chert gradually disappears. The large fragments of chert in the pits become sandy, soft and granular towards the surface, due to the leaching of silica, and ultimately almost entirely disappear.

b. Clay gives way to bauxite.

c. Limonite gives way to hematite and magnetite, as shown by analyses and color.

d. There is a breaking-down of the coarse fragments of chert and boulders of serpentine found in the ore, due to the leaching of constituents, resulting in fragments and granules of great variety of size and shape, in general diminishing in size towards the surface, and ultimately reaching an earthy texture.

Near the surface there is a local and distinct tendency for the development of granules from the extremely fine products of alteration. This seems to be a constructive process, distinguishing these granules from the large and irregular ones derived from the breaking-down of larger fragments beneath the surface.

In the development of the Mayari ores from the serpentine there is left a fine, powder-like mass, which is gradually reconstructed near the surface into granules. The Camaguey ores, on the other hand, contain large residual masses of chert, giving the mass quite a different texture. Also, boulders of serpentine in the conglomerate at the base of the limestone, while altering to a soft, powdery mass, distinctly retain their outlines, as observed in the open pits. Thus the Camaguey ore has decidedly a coarser and more irregular texture than the Mayari ore. Where alterations go to an extreme in the Camaguey ores, as they do near the surface, the texture may become as fine as in the Mayari ores, but this extreme is reached in a much smaller portion of the mass than in the Mayari deposits. The reconstruction of the ore into minute granules at the surface is also less conspicuous in the Camaguey deposits than in the Mayari deposits, for the reason that the granules in the Mayari deposits stand out in sharp contrast to a fine-textured mass of ore below, while in the Camaguey deposits they are in comparison and likely to be confused with the irregular fragments of chert and altered serpentine below.

Mechanical erosion has taken off the top of the ore to different levels. Where erosion has been slight and secondary processes of alteration, especially the leaching of silica, have therefore been allowed to work undisturbed, the grade of the ore is likely to be good. Where erosion has been rapid and deep, it may have cut down to the parts of the body containing large chert masses, exposing them at the surface; thus producing the local differences in character of the ore at the surface.

1. *Source of Camaguey Deposits.*

The Camaguey deposits have certain features in common with the better-developed Mayari deposits on Nipe bay. Both are mantle-deposits of somewhat similar mineralogical and chemical composition, resting on the surface of serpentine. The Mayari deposits have been demonstrated to be residual deposits resulting from the alteration of the serpentine in place. This would naturally suggest similar origin for the Camaguey deposits, yet certain facts suggest a possibly different origin, which may account for certain significant differences in composition.

The part of the Camaguey deposits examined covers an area of a plateau about 8 miles N–S. by about 10 miles E–W. Beneath the deposits is serpentine. To the south of the deposits is a plateau of serpentine from 30 to 60 ft. lower than the iron-ore plateau. Erosion has evidently stripped the iron-ore from this lower plateau, for residual fragments of iron-ore and chert cover the surface of the serpentine plateau. To the north the ore-deposits are bounded by overlying Cretaceous (?) limestone dipping northward and forming a high, northward-facing escarpment. It is apparent that the limestone has at one time covered a much wider area than at present, and, indeed, has probably covered all of the serpentine and ore area. The removal of this limestone may have left residual deposits of iron-ore. The alternative explanation is that the iron-ore is the direct result of the alteration of the underlying serpentine in place. The hypothesis of the derivation of the ore from the limestone rather than the serpentine seems to us to be favored by the following considerations:

a. The ore contains abundant and conspicuous chert fragments, especially near the bottom, which are common in limestone and which are known to accumulate in the residual deposits of limestone decay. Cherts (some of them radiolarian) are described by Hayes and Spencer as common in the Cuban limestone. The cherts themselves have a banded texture and solution-cavities, strongly suggestive of original interbedding with carbonates. On the other hand, the serpentine, so far as we observed it, contains no chert which could have yielded the chert now seen in the ore.

b. The ore is distinctly conglomeratic in texture near the bottom and contains large boulder-like masses, now composed of iron-ore, which were perhaps originally serpentine boulders in a conglomerate at the base of the limestone.

The nickel- and chromium-content of the Camaguey ores is more likely to have been derived from serpentine alteration than from limestone alteration, and it is suggested that their source may be the abundant altered serpentine boulders in this conglomerate.

c. The ores contain lime and magnesia, which are absent in the Mayari ores, known to be derived from the serpentine deposits. If the Camaguey ores are derived from serpentine, there is no reason why lime and magnesia should not be here also completely absent.

d. The Camaguey ore is higher in phosphorus than the Mayari ores derived from serpentine. High phosphorus is characteristic of residual deposits from limestone. The brown ores of the United States, largely of this class, illustrate this fact.

e. Iron-ore deposits are common residuals from limestone; in fact, in various parts of Cuba the weathering of limestone may be seen to yield red soils containing considerable percentages of iron.

It seems to us that the facts yet available do not warrant a choice between the two available hypotheses of the source of the Camaguey ores. The similarities of the Camaguey to the Mayari ores suggest their residual accumulation from the alteration of serpentine. The differences suggest the original residual accumulation of the Camaguey ores from a once-overlying limestone. If it should ultimately be found that the Camaguey ores are residual from limestone rather than serpentine, it follows that the lower contact should be the more or less uniform one, with regular drainage-channels, of an erosion-surface, upon which the limestone was originally deposited. Underground explorations have not yet gone far enough to demonstrate this. It is entirely possible that along certain deeper main drainage-channels the ore may be found to be deeper than the depths now known. The Mayari deposits, on the other hand, resulting from residual alteration of the serpentine, lack any regularity of contact or regular drainage-channels.

Occurrence, Origin, and Character of the Surficial Iron-Ores of Camaguey and Oriente Provinces, Cuba.

BY ARTHUR C. SPENCER, WASHINGTON, D. C.

(Wilkes-Barre Meeting, June, 1911.)

THREE great deposits of iron-ore, in Camaguey and Oriente Provinces, Cuba, are well known to me through careful field-examinations executed in the years 1901 and 1907.

In 1901 I visited the Cubitas iron-ore district, which lies about 12 miles distant from the city of Camaguey in a northerly direction, and the Mayari district, which includes the Sierra Nipe, lying opposite Nipe bay on the north side of Oriente Province. In 1907 I again visited the Cubitas district, and also made a sojourn of several days in the Moa district, where the extensive deposits of iron-ore were observed and studied.

The observations of 1901 were made under the auspices of the then Military Governor of Cuba, Gen. Leonard Wood, and my conclusions concerning the value of these deposits were incorporated in a report.[1]

The examination of certain denouncements of iron-ore in the Moa district in 1907 was made in behalf of iron-masters operating in the United States, to whom I reported the existence of large amounts of easily-workable limonitic iron-ore, properly designated "brown iron-ore" in the terminology now current among iron-ore producers in the United States.

In 1908 I published a paper entitled, Three Deposits of Iron Ore in Cuba, outlining the occurrence and origin of the surficial ores existing in the Cubitas, Mayari, and Moa districts.[2]

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[1] *A Geological Reconnoissance of Cuba*, by C. Willard Hayes, T. Wayland Vaughan, and Arthur C. Spencer, printed as a part of the report of the Military Governor of Cuba for the year 1901, vol. i.

[2] *Bulletin No.* 340, *U. S. Geological Survey*, pp. 318 to 329 (1908).

OCCURRENCE.

The vast tonnages of iron-ore existing in the Cubitas, Mayari, and Moa districts, in the island of Cuba, occupy the tops of flat or gently-sloping plateaus. The ores constitute surficial mantles over extensive areas of these plateaus, and in each district the deposits are underlain by serpentine rock. Within areas having an extent, in each district, of several square miles, the mantles of ore are essentially continuous over the surface of the ground, excepting where they have been eroded by running streams.

The manner in which these deposits of iron-ore occur, their attitude with respect to the serpentine rock upon which they lie, and the topographic features of the ore-fields, considered in connection with a comparison of the chemical composition of the ores with that of the serpentine, point definitely and unmistakably to the manner in which the ores in question have been formed.

In considering the surficial iron-ores of Cuba, it is of interest to note that iron-ores occur in very similar relationship in various parts of the world. Indeed, such ores have long been known, and in many places have been used as a source of pig-iron.

In the United States, iron-ores of precisely similar composition forming surface-deposits over serpentine rock exist at Clealum, Wash., and at Richmond, Staten Island, near New York City. The Staten Island deposits, though now practically exhausted, were formerly mined and smelted. The Clealum ores occur in a region in which smelting would not be profitable because no cheap fuel is available for this purpose.

Similar ores in the same relationship to serpentine rock occur also on the island of New Caledonia,[3] in Western Australia,[4] and in several localities adjacent to the Mediterranean sea.

Deep and extensive surface-mantles of iron-ore occurring in India closely resemble the Cuban ores here under discussion in physical character and in the fact that they occupy elevated plains or plateaus, though in India the underlying rock is basalt and not serpentine. These deposits have been mined and

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[3] E. Glasser, *Annales des Mines*, Tenth Series, vol. v., pp. 111 to 125 (1904).

[4] A. Gibb Maitland, *Annual Progress Report of the Geological Survey*, Perth, W. A., p. 22 (1905).

smelted for hundreds of years by the natives of India, and are now reported as being developed under government auspices.

From the facts above stated, it is evident that ores of the nature of those occurring in the Cubitas, Mayari, and Moa districts have been long established among the different varieties of iron-ores, and cannot be considered properly under any other classification.

Origin.

The manner in which the surficial iron-ores of Cuba were formed was first stated by me in a paper entitled, Three Deposits of Iron Ore in Cuba,[5] printed in 1908. The mode of origin which I have outlined is in accord with the findings of the Government Geologist of India, in regard to the origin of surficial ores of that country, which are described as high-level laterite and carry less iron and more alumina than the Cuban residual ores, but are undoubtedly of similar origin; also in accord with the conclusions of T. Sterry Hunt[6] on the origin of the Staten Island surficial ores, and again essentially in accord with Bailey Willis and George Otis Smith[7] on the origin of the iron-ores at Clealum, Wash.

My conclusions concerning the origin of the surficial iron-ores of Cuba may be briefly stated as follows: These ores have been formed as a result of progressive and long-continued decay of serpentine rock under the dissolving and corroding action of atmospheric waters charged with carbonic acid gas. The ores are thus properly termed residual ores in the sense that they have originated *in situ* through the gradual wasting of the serpentine rock, the removal of its more easily attacked constituents, such as magnesia and silica, by a process of solution, and the consequent setting free of oxides of metals such as those of iron and aluminum, which are well known to be practically insoluble in reagents ordinarily found in nature.

The parentage of the surficial ores of Cuba in serpentine rocks like those upon which they lie is conclusively established by the presence of small percentages of chromium, nickel, and cobalt in the ores. These metals are characteristic constitu-

[5] *Bulletin No.* 340, *U. S. Geological Survey* (1908).

[6] *Mineral Physiology and Physiography*, pp. 268 to 269 (1886).

[7] *Trans.*, xxx., 356 to 366 (1900).

ents of serpentine rocks, as shown by analyses of serpentines collected in various parts of the world. Their presence in appreciable amounts has been established in the serpentines of Cuba and in similar rocks lying beneath the previously mentioned Staten Island and Clealum ores. The residual ores of these localities carry chromium and nickel as do those of Cuba.

I am informed by G. M. Colvocoresses that surficial iron-ores occurring in New Caledonia which carry small amounts of chromium, nickel, and cobalt, were formed in exactly the manner which I outlined in my 1908 report.

The ores in question cannot be otherwise classified than as residual ores. That is, they represent the insoluble residue left by otherwise-complete dissolution of serpentine rock under the action of atmospheric waters. This mode of origin has been fully discussed and accepted by C. M. Weld.[8] They are to be set apart from that other class of surficial iron-ores known as bog ores, since the well-known origin of the latter is very different. Bog iron-ores are deposited in swamps and marshes from dilute solutions in which the solvent is ordinarily either an organic acid derived from decaying vegetation, or sulphuric acid produced by oxidization of iron pyrites. The precipitation from such solutions is known to be effected by micro-organisms which inhabit the waters of swamps and marshes, or by carbonization under reducing conditions in the presence of decaying organic matter. Residual ores like those of Cuba and India are deposited *in situ*, while bog ores, which are characteristically of very limited extent, are mainly deposited at some distance from the source of the contributory iron, which involves transportation to the place of deposition in a dissolved condition. The residual ores of Cuba were formed in Tertiary time, in large part, and perhaps entirely, prior to the deposition of the Lafayette (Pliocene) formation of the Atlantic coastal plain. During the time of their accumulation the brown iron-ores of Alabama, Virginia, Pennsylvania, and New York were being deposited, many occurrences of which are likewise recognized by geologists as being residual ores.[9]

[8] The Residual Brown Iron-Ores of Cuba, *Trans.*, xl., 299 to 312 (1910).

[9] Edwin C. Eckel, *Bulletin No.* 400, *U. S. Geological Survey*, pp. 145 to 150 (1910).

Character.

The character of the residual iron-ores of Cuba is in general similar to that of residual iron-ores formed during the same geologic period in the eastern part of the United States, and in particular almost precisely like the character of certain iron-ores occurring at Richmond, Staten Island, N. Y. The Cuban ores in question consist in large part of extremely hard round pellets and irregular nodules, often nearly black in color. The pellets vary in size from that of a pin-head up to that of a cherry, while the nodules range up to a diameter of several inches. Both are commonly imbedded in earthy material, which ordinarily has essentially the same composition as the hard portions of the aggregate and, like them, is iron-ore. In many places hard ore free from matrix of earth forms solid layers, evidently of very considerable extent. I have noted such layers 4 m. thick in the Moa district.

Analysis shows the presence of water of constitution in these ores, that is, combined water which is not expelled by heating the ore to a temperature of 100° C. Such analyses as have been made for me indicate that the amount of this combined water is less than that required by the mineral species limonite, and very considerably less than this requirement, if the rather high alumina-content of the ores be considered as present in the form of the hydrated oxide corresponding to the ordinary ore of alumina, bauxite. The inference follows that these ores carry part of their iron in the form of unhydrated ferric oxide. Not only this, but part of the ore in its natural undried condition possesses the quality of being drawn by a magnet, a characteristic independently distinguishing it from true limonite. Certain samples of shot or pellet ore which I collected in the Mayari district contain approximately 5 per cent. of material which may be separated by means of an ordinary pocket magnet. A sample of such ore tested at the Newark works of the Wetherill Separating Co., in 1902, was found to contain no non-magnetic material. The ore was crushed to pass 20 mesh, and with successive strengths of field corresponding to currents of 1, 2, 3, and 4 amperes, yielded four products amounting respectively to 20, 33.3, 33.3, and 13.3 per cent. of the material treated.

Considered from all points of view, the Cuban residual ores must be assigned to the limonitic class, but they do not strictly conform to any member of the limonite group and it has been found convenient to call them brown ores, to cover the impracticability of any distinctive varietal name. Attention is called to their dissimilarity with the variety of limonite known as bog ore. In addition to the fact that their mode of origin was entirely different, they show a uniformly low tenor of phosphorus, which separates them absolutely from bog ores, which are characteristically high in phosphorus-content.

The foregoing discussion relates essentially to the upper portion of the ferruginous mantles in the three districts which I have had opportunity to visit and study. My characterization of the Cubitas, Mayari, and Moa ores as brown ores was made from general observations, extended over considerable areas in each field, but the conditions under which my examinations were made did not admit the making of excavations. For this reason, I have not been able to indicate completely from personal observations the progressive change in physical character from the surface of the ground downward through the ferruginous residuum to the undecomposed serpentine rock. The development-work carried on by the Spanish-American Iron Co. in the Mayari district is reported to have shown that the upper part of the ore-bed is underlain in many places by red or yellow ore of a clay-like consistency. The technical point has arisen whether or not this material, and the brown ore occurring as lumps or pellets, can be mined independently of one another. Upon this question I may say that during parts of two days (1901) spent in traversing the Sierra Nipe plateau (Mayari district) I found no exposures of clay-like ore, and made the note that the brown ore formed a practically continuous mantle over the plateau in all parts visited. The thickness of the ore was judged to vary from 3 to 15 ft. over an area of many square miles.

In traversing the entire ore-field of the Cubitas district on two separate occasions, I did not observe any exposures of clay ore. In my detailed examination of the Moa district I saw such material at only one locality. This was at a shaft judged to be about 50 ft. deep. At this shaft the surface of the ground is covered by the usual brown ore of the district. From these

observations it is my best judgment that such clay ore as may exist in the three districts here under consideration, certainly lies beneath the brown ore and cannot be mined without disturbing the latter. As a matter of economy, the two sorts of ore should be mined together.

Summary.

The ferruginous deposits of the Cubitas, Mayari, and Moa districts, Cuba, occur as surficial mantles covering extensive plateau-like areas underlain by serpentine rock. The material of these deposits is brown iron-ore of residual origin formed in place by the chemical disintegration of the serpentine. The ores are limonitic in character, but are not true limonite, since they carry a certain amount of iron oxide uncombined with water. They are not bog ores, because their mode of origin and low tenor of phosphorus preclude this classification. I have preferred to call the ores simply brown ores. In localities where clay-like ore exists, it lies beneath the brown ore and cannot be mined without disturbing the latter.

The Mayari and Moa Iron-Ore Deposits in Cuba.

BY C WILLARD HAYES, WASHINGTON, D. C.

(Wilkes-Barre Meeting, June, 1911.)

The determination of the question whether the Mayari and Moa mining-claims of the Spanish-American Iron Co. have been rightly denounced under the third section of the law of bases rests on the findings in the following questions of fact:

1. Is the mineral an iron-ore?
2. Is the iron-ore a bog iron-ore?
3. Is the iron-ore ocher?
4. If ocher is present, can it be mined separately and independently of the iron-ore?

1. Is the Mineral an Iron-Ore?

Since the material is shown by a large number of analyses to contain from 41 to 50 per cent. of metallic iron and less than

0.02 per cent. of phosphorus, and since it has been and is being actually used on a commercial scale for the production of iron and steel, it must be classed as an iron-ore.

2. Is the Iron-Ore a Bog Iron-Ore?

Bog iron-ore has certain invariable characteristics of chemical composition, physical appearance, geological relations, and origin by which it can always be recognized with certainty.

In chemical composition, it is the hydrated sesquioxide of iron, limonite, or a mixture of limonite and other closely-related iron hydrates. It never contains the anhydrous oxides—hematite or magnetite; is never magnetic, and never contains either nickel, chromium, or cobalt oxides. On the other hand, it is invariably high in phosphorus-content.

In physical appearance and texture it is a yellow or reddish-brown amorphous spongy material, and always contains water-worn sand-grains, silt, clay, and plant-remains.

In its geological relations it is wholly independent of the rock on which it rests, and the character of the underlying rocks has no influence whatever on the physical character and mineralogical composition of the ore.

In origin it depends on (1) the solution of iron-minerals widely disseminated through the rocks, with the formation of ferrous salts with organic acids; (2) the transportation of these ferrous salts by running water; (3) their collection in swamps or ponds; and (4) the precipitation of the ferric hydrate through the oxidation of the easily decomposed ferrous compounds.

In all iron-ore deposits except magnetite there is more or less solution of the iron by percolating acidulated waters, but the iron is almost immediately redeposited in the same locality from which it was derived, frequently cementing into a solid mass the other portions of the same deposit. Springs issuing from iron-ore deposits generally hold a large amount of iron in solution, and this is deposited at the point of issue. Such solution and redeposition of a pre-existing iron-ore deposit does not form bog ore.

The Mayari and Moa iron-ores differ radically in all of these essential characteristics, namely: They consist of a mixture of hydrated iron oxide, limonite, with hydrated aluminum oxide,

bauxite, and the oxides of nickel, chromium, and cobalt, and are to some extent magnetic.

In physical appearance the ore is a reddish-yellow powder in which are imbedded, most abundantly in the upper portion, fine, shot-like concretions of darker reddish-brown or black color. These concretions are in places concentrated into a more or less compact mass with distinctly oölitic structure.

In its geological relations the ore is distinctly related to the rock on which it rests in the form of a mantle. It is confined exclusively to areas underlain by a particular type of altered igneous rock—serpentine. It is never found resting on limestones, sandstones, or shales, which occur abundantly in the eastern provinces of Cuba.

In its origin the Mayari and Moa iron-ore is undoubtedly derived directly from the underlying serpentine by the process of weathering, through which certain of the constituents of the rock have been removed in solution and the remaining constituents have been oxidized, hydrated, and concentrated practically *in situ*. The genetic relation between the underlying rock and the overlying ore is shown by the analyses of rock and ore given in Table I.

TABLE I.—*Analyses of Mayari Iron-Ore and Underlying Rock.*

	SiO_2.	Al_2O_3	Fe.	Mg.	Cr, Ni, Co.	Ratio $\frac{Fe}{Al_2O_3}$.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	
A. Rock underlying iron-ore (average of 3 analyses)......	39.60	1.61	7.04	20.07	1.50	4.37
B. Mayari iron-ore (average of 59 analyses)................	3.72	9.63	47.60	none.	2.95	4.83
C. Ratio between constituents in the rock and in the ore......	0.094	5.98	6.76	....	1.97	

It will be noted that the ore contains no constituent which is not also present in the rock; that the change from rock to ore consists in the complete removal of the magnesium, the nearly-complete removal of the silica, and the partial removal of the nickel, chromium, and cobalt, while the iron and alumina have retained nearly the same ratio in the ore as in the original rock.

The above comparison proves conclusively that the Mayari and Moa iron-ore is not a bog iron-ore, and therefore is correctly placed in the third section.

3. Is the Mayari and Moa Iron-Ore Ocher?

Yellow ocher, which is the only kind that requires consideration here, has no definite, fixed chemical and mineralogical composition and hence cannot be defined with scientific exactness. Its definition is commercial rather than chemical or mineralogical, but the name can be applied only to materials having essentially the same chemical and mineralogical composition as the well-recognized commercial ochers. It cannot be applied to material of radically-different composition, even though such material might possibly be used as a substitute for ordinary ocher.

The essential constituents of yellow ocher are hydrated ferric oxide (limonite) and clay (aluminum silicate). Since the clay is invariably present in all commercial ochers, even the best grades, it must be considered an essential constituent and not an accidental impurity.

Table II. indicates the wide range in composition of commercial yellow ocher.

Table II.—*Analyses of Commercial Yellow Ocher.*

	Locality and Authority.	Fe_2O_3	Al_2O_3.	SiO_2.	MnO_2.	H_2O.	Al_2O_3, $2SiO_2$ (Clay).	Excess SiO_2.
		Per Cent	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
1	Cartersville, Ga., Watson, av. 8 analyses............	63.43	7.05	16.69	1.09	9.84	15.36	8.36
2	East Whately, Mass., Shepard, av 3 analyses..	56.10	4.87	26.67	....	11.83	10.61	21.26
3	Cartersville. Ga., Merrill.....	55.84	....	....	....	12.00	32.20	
4	Keegletown, Rockingham Co., Va., Campbell......	52.28	....	....	....	6.35	40.22	
5	Northampton Co., Pa., Merrill......................	42.45	....	....	....	11.85	30.58	
6	Marksville, Page Co., Va...	39.00	15.00	33.00	....	11.50	32.70	15.30
7	Hancock, Pa., Merrill........	36.67	....	....	....	10.60	50.00	
8	Warren Co, Va., McCreath,	34.00	15.51	31.64	6.20	8.87	29.45	15.70
9	Topton, Pa., Stoddard.......	17.49	18.66	55.50	....	8.35	40.67	33.48
10	Montgomery Co., Ala., Merrill......................	10.57	....	....	....	7.40	32.20	
11	Breiningsville, Pa., Merrill,	9.27	17.40	60.53	....	5.51	20.21	49.59

From Table II. it is seen that the content of ferric oxide in yellow ocher varies from a maximum of 63.43 down to 9.27 per cent.; that the combined silicate of aluminum, that is, clay, varies from a minimum of 10.61 up to 50 per cent., and that there is generally an excess of free or uncombined silica.

In order to afford a basis for comparison of commercial yellow ocher and the Mayari and Moa iron-ores, Table III. is presented.

TABLE III.—*Analyses of Mayari and Moa Iron-Ore.*

	Depth	Analyses Averaged	Fe.	Fe_2O_3	SiO_2	Al_2O_3	Ni, Cr, Co.	P.	Comb H_2O.	Excess Al_2O_3.
	Feet.	No	Per Cent	Per Cent.	Per Cent	Per Cent.	Per Cent	Per Cent.	Per Cent	Per Cent.
Mayari pit 5.....	0–19	19	45.67	65 31	6.26	10.64	2.80	0.008	11.59	5.33
Mayari pit X.....	0–21	21	48.01	68.65	2.35	9.18	3.23	0.010	13.30	7.18
Mayari pit 8.....	0–19	19	49.12	70.24	2 56	9.06	2.82	0.007	12.01	6.88
Moa F., 1906....	0–36	7	41.27	59.02	3.01	14.57	1.91	0.008	....	12.01
Moa H., 1906..	0–16	4	41.10	58.77	5 64	12.04	1.79	0.011	....	7.25
Moa I., 1906.....	0–11	3	44.46	63.58	5.53	13.41	1.59	0.012	....	8.71
Moa F. 1, 1906..	0–24	5	44.73	63 96	4.04	13.46	2.01	0.005	....	10.03
Moa G. 1, 1906..	0–48	7	44.35	63 42	2 23	14.70	1.51	0.006	....	12.81
Moa F. 5, 1906..	0–24	6	45.16	64 57	3.09	12.07	1.45	0.008	....	9.44
Moa F. 6, 1906..	0–37	7	48.23	68.97	1.97	9.43	1.52	0.010	....	7.75
Moa G. 6, 1906..	0–20	6	45.48	65.04	3.33	11.58	1.40	0.007	....	8.75
Moa E. 5, 1906..	0–24	5	44.65	63.84	3.44	10.68	1.53	0.011	....	7.74
Moa E. 6, 1906..	0–12	3	46.22	66.09	3.73	12.23	1.36	0.011	....	9.06

Table III. shows that the content of ferric oxide, Fe_2O_3, in the Mayari and Moa iron-ore varies from a minimum of 58.77 up to and beyond 70 per cent. By comparison with the analyses of ocher it is seen that ocher from only a single locality, Cartersville, Ga., contains as much ferric oxide (Fe_2O_3) as the lowest-grade ore from Mayari and Moa. All of the ochers from other localities contain less iron than the minimum permissible in an iron-ore.

It is further noted that the Mayari-Moa iron-ores contain alumina, Al_2O_3, in excess of the amount required to combine with the silica, SiO_2, present to form clay, Al_2O_3, 2 SiO_2. This excess of alumina varies from 5.33 to 12.81 per cent., and it is undoubtedly present as the mineral bauxite, having the empirical formula Al_2O_3 (Fe_2O_3, SiO_2, H_2O). The Mayari-Moa ores must therefore be considered as made up of an intimate mixture of:

(1) limonite, Fe_2O_3, H_2O;

(2) bauxite, Al_2O_3 (Fe_2O_3, SiO_2, H_2O);

(3) chromite, $FeCr_2O_4$, and some undetermined compound of nickel and cobalt.

It is evident from the above comparison that commercial yellow ochers and the Mayari-Moa iron-ores are entirely distinct chemically and mineralogically and radically unlike.

4. If Ocher is Present, Can It be Mined Separately and Independently of the Iron-Ore?

The iron-ore occurs at Mayari and Moa in the form of a mantle or blanket overlying the serpentine, from which it has been derived by the process of rock-weathering. It varies in thickness from a few inches to 50 ft. or more, this variation depending upon the varying rate at which the processes of rock-weathering have acted, but more directly on the varying rate at which the products of alteration have been removed by surface-erosion. The variation with depth consists chiefly in the decrease downward in proportion of concretionary "shot" ore to the fine pulverulent ore. The material, however, is a commercial iron-ore from the upper surface of the deposit down to the surface of the unaltered rock. There is no over-burden, and no gangue which must be separated in mining the ore.

It has been shown above that the ore is not ocher, but if it should be demonstrated that some portions of the deposits might be utilized for the same purposes for which yellow ocher is used, the analyses show that these very portions are the ones most valuable as iron-ore. Hence the utilization of the deposits as a substitute for ocher precludes their utilization to that extent as iron-ore.

Yellow ocher is used in the arts for two purposes chiefly: as a filler for oil-cloth and as a pigment. For these purposes, after mining, it is subjected to an expensive process of washing, by which all sand and other objectionable ingredients are removed, leaving in the prepared material only iron oxide and clay. This preparation for market is the most important element in its cost. Independently of cost the market for ocher is very limited, as shown by Table IV.

TABLE IV.—*Production of Ocher and Iron-Ore in the United States.*

	Ocher.	Iron-Ore
	Long Tons.	Long Tons
1904	16,826	27,644,330
1905	13,402	42,526,133
1906	15,482	47,749,728
1907	16,971	51,720,619
1908	17,019	35,983,336

For the five years, 1904–1908, there was mined in the United States 2,580 tons of iron-ore for each ton of ocher. Since the production of ocher was determined by the demand for use in the arts rather than by any limitation of the deposits from which it was derived, it is evident that to secure the ocher market Cuban material would have to displace that from some other source. Even if the entire market in the United States could be secured and supplied by the Cuban deposits, it would require but a small portion of the yield which these deposits are capable of producing if the material continues to be used as an iron-ore. Moreover, the demand is not sufficient to warrant the building and maintenance of railroads, docks, and steamship-lines merely for the ocher trade, and without these appliances the expense of mining, preparing, and transporting the ocher to market would be so great as to be prohibitive in competition with deposits more favorably located with reference to transportation-facilities and markets.

The conclusion appears necessary, therefore, that even if a portion of the material in the Mayari-Moa iron-ore deposits might be substituted for yellow ocher, it could not be mined separately and independently of the iron-ore, and it could not be mined at a profit exclusively as a substitute for ocher.

Characteristics and Origin of the Brown Iron-Ores of Camaguey and Moa, Cuba.

BY WILLARD L. CUMINGS * AND BENJAMIN L. MILLER,† SOUTH BETHLEHEM, PA.

(Wilkes-Barre Meeting, June, 1911)

I. The Camaguey Deposits.

1. *Location.*

The Camaguey brown iron-ore deposit covers the top of San Felipe hill, the nearest point of which lies 14 miles NW. of the city of Camaguey. While there are several low flat-topped hills in the vicinity covered with a more or less continuous mantle of brown iron-ore, the deposit of San Felipe hill is the only one of any size and importance, and the name "San Felipe district" is proposed for the region.

The deposit extends in a NW–SE. direction for a distance of about 10 miles, with an average width of 5 miles. The location is shown in Fig. 1, a sketch-map of the eastern part of Cuba. Fig. 2 is a map of the San Felipe district.

2. *Description.*

The district is mentioned by Spencer,[1] who says:

"The Cubitas iron-ore fields are situated from 12 to 15 miles north of Camaguey City, in the province of Camaguey. . . . Within an area measuring roughly 10 miles east and west and 4 miles north and south, there are several flat-topped mesas rising 300 to 400 ft. above the level of an almost featureless plain which extends for many miles in all directions except toward the north."

With the exception of the discrepancy in his statements as to the distance from Camaguey and the area of the hill, which is over 50 sq. miles, his description is a very good one.

Kemp[2] has the following to say of ores in Camaguey Province:

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* Geologist, Bethlehem Steel Co.

† Professor of Geology, Lehigh University.

[1] *Bulletin No. 340, U. S. Geological Survey,* p. 324 (1908).

[2] *The Iron Ore Resources of the World,* printed by the Eleventh International Geological Congress, vol. ii., p. 795 (1910).

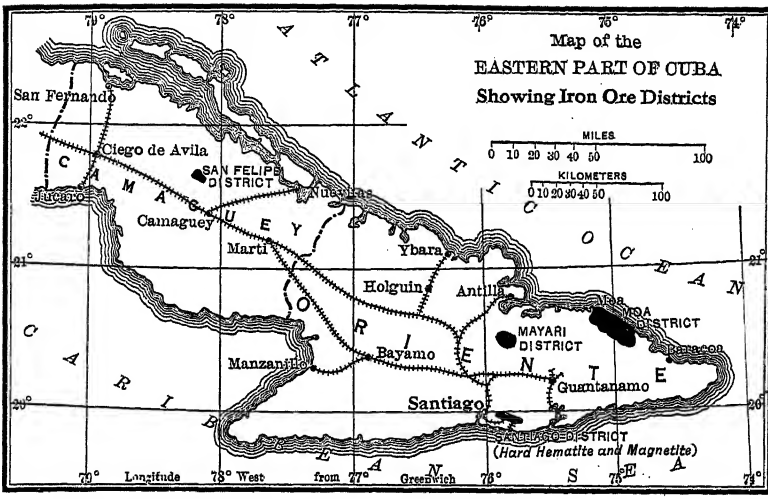

FIG. 1.—SKETCH-MAP OF THE EASTERN PART OF CUBA, SHOWING IRON-ORE DISTRICTS.

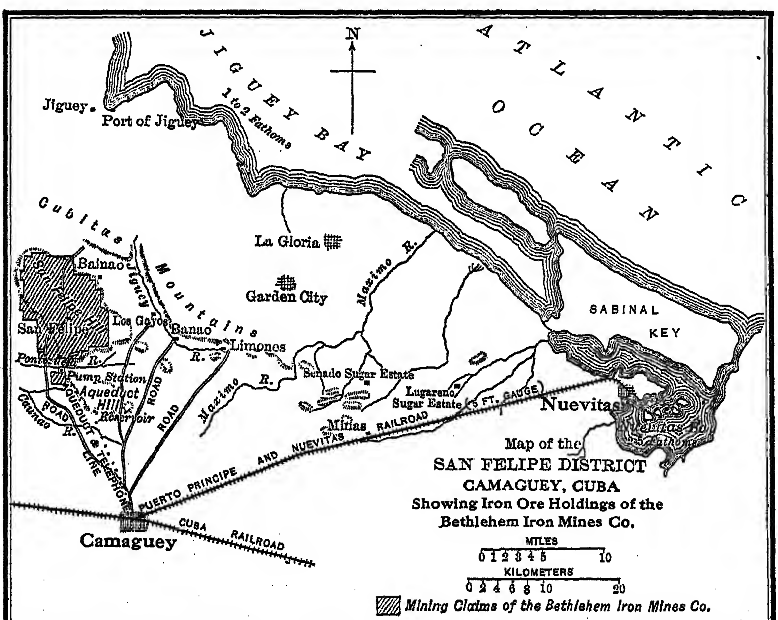

FIG. 2.—SKETCH-MAP OF THE SAN FELIPE IRON-ORE DISTRICT, CAMAGUEY, CUBA.

"Similar surface materials [to those of Moa and Mayari] have been tested at Cubitas in Camaguey province, but extended borings failed to show anything richer than 30 per cent. in iron."

Since Dr. Spencer referred to this district as "the Cubitas field," one would suppose that Kemp refers to the same deposit. As far as we are aware, however, there is no definite locality known as Cubitas, and, as will be shown further on, the iron-content he mentions is about two-thirds of what San Felipe ores show. Probably he refers to some point in the Cubitas mountains, where some exploration has been done in the lean-ore mantle overlying the limestones. This occurrence will also be referred to later.

3. *Topography.*

The topographic features of San Felipe are extremely simple. Barometer-readings at Pontezuela river, south of San Felipe hill, show elevation of 190 ft. above sea-level, while the whole top of the hill varies from 450 to 500 ft. While the south and east sides are quite steep, the NW. part of the hill has gradual slopes, thus affording an easy ascent for a railway. North of the hill is the valley of the Jiguey river, and beyond this is the escarpment of the Cubitas mountains facing south. San Felipe hill and almost all of the surrounding country, except the limestone-areas of the Cubitas mountains, are covered by small stunted palms, indicative of poor soil. There are, however, considerable detached areas, called *montes*, which support the dense growth of small timber common to Cuba.

4. *Geology.*

Like the ores at Moa and Mayari, the San Felipe ores occur as a mantle on serpentine or altered peridotite. It is probable that the area of serpentine near Camaguey is very large indeed. A few miles north of Camaguey the peridotite and occasional pebbles and boulders of the iron-ore are noticed. The country-rock is also serpentine for several miles along the line of the Puerte Principe & Neuvitas railroad. We are informed by R. L. Luaces, of Camaguey, that some years ago a deep well put down in Camaguey passed through several hundred feet of peridotite and then entered granite. On the north the serpentine is not seen beyond the Jiguey river, and to the

west it disappears about half way to the John Fritz mine, a deposit of magnetic hematite located 22 km. west of San Felipe and which occurs in diorites and syenites similar to the mines at Santiago.

The serpentine differs in no essential respect from that at Moa. On the lower slopes of the hills, where erosion has removed the soft decomposition-products, and along streams, it is fresh and unaltered and the large crystals of pyroxene show plainly in some places. The only product of alteration that is out of the ordinary is the large amount of chert, which is in the form of fragments and which is especially noticeable on the surrounding hills, where the iron-ore undoubtedly present at one time has subsequently been removed by erosion. Thus, on Aqueduct hill, there is very little ore to be seen, but the rough surface is covered with a mixture of chert and serpentine that in weathering has formed rough sharp projections that make traveling very difficult. Chert of similar character has been noted in many other regions where peridotite has been altered to serpentine.

North of San Felipe, the contact between the serpentine and the Triassic limestone is very sharp and apparently follows the course of the Jiguey river. The limestone is a typical massive white limestone and presents few good exposures where the dip can be determined. However, in a gorge north of Limones, the dip seems to be about 50° south, indicating the possibility that the limestone dips under the serpentine.

The limestone forms the south-facing escarpment of the Cubitas mountains. From the top of the escarpment the country has a gradual northerly slope to sea-level.

Over the whole Cubitas mountain limestone-area there is more or less red lean iron-ore somewhat similar in appearance to the San Felipe ore. There is, however, very little, if any, hard ore, and the occurrence of shot ore is rare. A sample of this ore from 3 km. east and 2 km. north of Banoa gave the following analysis: Fe, 34.57; Mn, 0.86; P, 0.055; S, 0.029; SiO_2, 12.11; Ni, 0.46; Cr, 1.33; loss on ignition, 8.77 per cent.

This lean-ore mantle is not absolutely continuous, and occasionally one may ride for a considerable distance over areas of exposed limestone. At one place between Banoa and La Gloria

there is an area where the weathering of this rock has produced curious forms. On either side of the trail are domes, spires, and tablets of various shapes, rising to heights of from 8 to 12 ft., and between them are bowl-shaped holes of solution, some of which are several feet in depth.

The mantle of lean iron-ore which overlies the limestone differs from either the San Felipe or Moa ores only in being lower in iron-content, and more pulverulent in character. In appearance it is simply a very red soil, and that it is fertile is shown by the abundant forest-growth in the Cubitas mountains. Its analysis shows that it is chemically similar to the ores overlying and derived from the serpentine in that it contains a large amount of alumina and both chromium and nickel. It is extremely improbable that the ore owes its origin entirely to the residual decay of the limestones, and we believe that, notwithstanding its existence as a mantle on limestone, its origin is only to be explained by derivation from the serpentine originally. There are indications that the Cubitas mountain escarpment was formed by a fault that is now followed by the Jiguey river.

5. *Characteristics of the Ore.*

Practically every one of the *mesas* in the San Felipe district contains a mantle of brown ore, and principally at an elevation of from 400 to 500 ft. above sea-level. On the smaller hills, however, erosion has proceeded so far that the ore is nearly all removed. In different parts of the plain, which has an elevation of from 150 to 250 ft. above sea-level, there is some ore and some mining-denouncements have been made, but the ore on these flats, or *sabanas*, is very shallow, and outcrops of serpentine appear at frequent intervals.

On the San Felipe hill there is a great deal of hard ore similar to that on the beach at Moa, and in places the boulders are of enormous size, as shown in Figs. 3 and 4. Over other areas, especially the wooded ones, there is no float ore, and the presence of the ore-deposit is only revealed by digging through the soil and vegetable matter, which is generally only a few inches deep. Fig. 5 illustrates the flat character of San Felipe hill, and Fig. 6 is a view of an ore-pit, showing the partly-disintegrated character of the surface-ore. An idealized section of the ore-mantle, showing average depths, is shown in Fig. 7.

Fig. 3.—Boulders of Iron-Ore on the Northeast Side of San Felipe Hill.

Fig. 4.—Boulders of Iron-Ore on the Northeast Side of San Felipe Hill.

FIG. 5.—VIEW SHOWING FLAT CHARACTER OF SAN FELIPE HILL.

FIG. 6.—PIT IN CENTER OF SAN FELIPE HILL, SHOWING PARTLY-DISINTEGRATED CHARACTER OF SURFACE-ORE.

The greatest difference between San Felipe and Moa and Mayari is the coarse nature of the disintegrated capping at the first-mentioned locality and the frequent presence of hard ore below. Thus, at San Felipe, some pits can be dug 30 ft. without the use of dynamite, while others can be dug only a few feet before the hard layer, necessitating blasting, is encountered, and in still other areas the hard ore is found immediately under the grass-roots. In no case has it been found possible to ex-

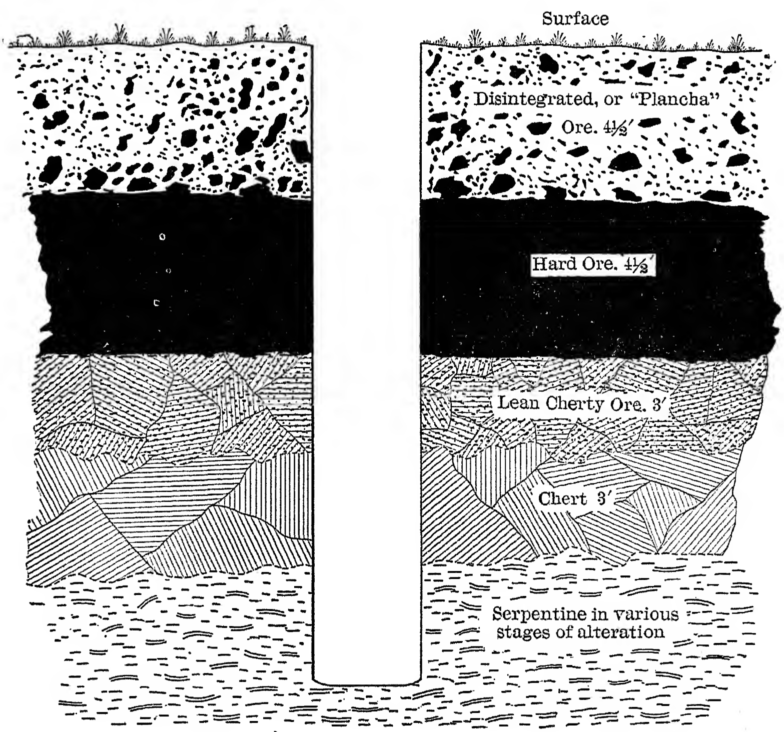

FIG. 7.—IDEAL SECTION OF TEST-PIT, SAN FELIPE DISTRICT.

plore with hand-augers, as was done at Moa and Mayari, as the auger is so apt to hit boulders of hard ore, that are frequently of considerable size.

The characteristic "shot ore," so well described by Weld[3] and Spencer,[4] is present over large areas.

A typical analysis of San Felipe ores is, in average of 10

[3] *Trans.*, xl., 299 to 312 (1910).

[4] *Bulletin No.* 340, *U. S. Geological Survey* (1908).

samples: Fe, 45.18; SiO_2, 6.75; Al_2O_3, 12.3; Mn, 0.56; Cr, 1.7; Ni, 0.53; P, 0.1; S, 0.063; CaO,MgO, 2; loss on ignition, 12 per cent.

This analysis shows that the San Felipe ore was evidently not the ore referred to by Kemp in his quotation in *The Iron Ore Resources of the World.*

A comparison of the average analysis given above with the following average analysis of three samples selected at random from a large number of Buena Vista (Moa) ores, shows the striking similarity of the ores of the two districts: Fe, 44; SiO_2, 1.62; Al_2O_3, 11.61; Mn, 1.18; Cr, 1.42; Ni, 0.76; P, 0.006; S, 0.332; CaO,MgO, 1.66; loss on ignition, 19.18 per cent.

It will be noted that in all respects, and even in the percentages of CaO and MgO, the agreement is so perfect that it would, indeed, be dangerous to assume from this a limestone origin for one and a serpentine origin for the other, as has been done by one recent investigator.[5]

The higher percentage of phosphorus in the San Felipe ores probably proves nothing, as ores of a similar origin vary in this element the world over.

The analyses quoted in this section were made by R. E. Kresge, Chemist, Bethlehem Steel Co., South Bethlehem, Pa., and by W. W. Fitch, Chemist, Bethlehem Iron Mines Co., Camaguey, Cuba.

6. *Economics.*

Not enough exploration has yet been done to prove the economic possibilities of the San Felipe iron-ores. Pits in 40-per cent. ore are common over the whole area of San Felipe hill. Certain pits have shown the following occurrences:

8 ft. of 41-per cent. ore.
26 ft. of 40-per cent. ore.
6 ft. of 43-per cent. ore.
18 ft. of 42-per cent. ore.

Other areas seem to indicate the presence of good tonnages of 45-per cent. ore and better, as the following pits show:

5 ft. of 45-per cent. ore.
3 ft. of 48-per cent. ore.
11 ft. of 46-per cent. ore.
7 ft. of 47-per cent. ore.

[5] Leith, this volume, p. 101.

Some areas have yielded 50-per cent. ore but, so far, no great amount of such ore has been found.

Judging from the enormous area controlled by the Bethlehem Iron Mines Co. (nearly 60 sq. miles), and assuming that one-third of this area is worthless, which makes an extremely conservative estimate, it is probable that there are 400,000,000 tons of 40-per cent. ore and 50,000,000 tons of 45-per cent. ore.

Some experiments already performed seem to show possibilities of raising the percentage of iron in the ore by screening or washing. Thus a pit showing a depth of 8 ft. of 41-per cent. ore was sampled and the sample screened through 0.25-in. screen. It was not sifted through, but simply thrown on an inclined screen exactly as mortar-sand is screened. Seventy-five per cent. of the sample proved to be coarser than 0.25 in. and this analyzed: Fe, 44; Al_2O_3, 12.30; SiO_2, 6.75 per cent. The 25 per cent. of fines or waste showed, Fe, 33; Al_2O_3, 17; SiO_2, 16 per cent. Washing this ore gave slightly better results. Careful experiments on 100-lb. samples of varying percentages of iron, but all above 40 per cent., seem to prove conclusively that simple screening will give a concentrate which will average 46 per cent. of iron, and which will not be finer than $\frac{1}{8}$-in. mesh. This will, however, be attended by considerable loss of fines, probably 40 per cent., which will be very high in alumina and silica.

Other economic features of the San Felipe deposit, aside from composition and possible mechanical enrichment, are most favorable. San Felipe, being less than 500 ft. above the sea-level and with gradual slopes on the west and north sides, requires no inclined planes. The ore, especially if screened, certainly needs no nodulizing to improve its physical character for furnace-use, and the known depths of ore and its coarse granular nature favor the work of steam-shovels. Also, as far as our observations have gone, the climate on the *mesas* as low as 500 ft. in elevation is somewhat better than on those at higher altitudes.

II. The Moa Deposits.

The Moa occurrence of brown ores has been so fully described by Spencer,[6] and especially by Weld,[7] that but slight

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[6] *Bulletin No.* 340, *U. S. Geological Survey* (1908).

[7] *Trans.*, xl., 299 to 312 (1910).

mention is necessary to show the close resemblance to the San Felipe occurrence.

There is on the shore of Moa bay a considerable outcrop of hard ore of the same pseudo-conglomeratic character as the San Felipe ore. Wherever rocks are exposed they are seen to consist of serpentine in various stages of alteration from the peridotite. Passing back from the coast at Moa, there is a gradual ascent and the rocks are hidden by a mantle of soft ore, sometimes 60 ft. deep. The occurrence of shot ore at the actual ground-surface is very common.

III. Proof that the Cuban Brown Ores of Camaguey and Moa Are Not Bog Ores.

1. *Definitions and Descriptions of Bog Ore.*

The description of bog ore by Sir Archibald Geikie,[8] the foremost English geologist, is as satisfactory as any that can be given.

"Bog Iron-Ore (Lake-ore, minerai des marais, Sumpferz)—a dark-brown to black, earthy, but sometimes compact mixture of hydrated peroxide of iron [limonite], phosphate of iron, and hydrated oxide of manganese, frequently with clay, sand, and organic matter. An ordinary specimen yielded, peroxide of iron [hematite], 62.59; oxide of manganese, 8.52; sand, 11.37; phosphoric acid, 1.50; sulphuric acid, traces; water and organic matter, 16.02 = 100 00 . . . . According to Ehrenberg, the formation of bog ore is due, not merely to the chemical actions arising from the decay of organic matter, but to a power possessed by diatoms of separating iron from water and depositing it as hydrous peroxide [limonite] within their siliceous framework." (p. 187.)

"Again, in the formation of extensive beds of bog-iron-ore, the agency of vegetable life is of prime importance. In marshy flats and shallow lakes, where the organic acids are abundantly supplied by decomposing plants, the salts of iron are attacked and dissolved. Exposure to the air leads to the oxidation of these solutions, and the consequent precipitation of the iron in the form of hydrated ferric oxide [limonite], which, mixed with similar combinations of manganese, and also with silica, phosphoric acid, lime, alumina and magnesia, constitutes the bog-ore so abundant on the lowlands of North Germany and other marshy tracts of northern Europe." (p. 612.)

Dr. Richard Beck,[9] Professor of Economic Geology in the Freiberg Mining Academy, and one of the foremost German geologists, described bog ore as follows:

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[8] *Text-book of Geology,* 4th (last) ed. (1906).

[9] *Lehre von den Erzlagerstätten* (Weed's translation, *The Nature of Ore Deposits*), p. 99 (1905).

"Bog iron ore, also called swamp ore, meadow ore and bog ore, is yellowish, brownish or blackish limonite, with resinous luster on fresh fractures, always highly porous and cavernous, often slag-like and hard, sometimes ochrous, loose, earthy and mingled with many other substances. The ores contain hydrated iron silicate (a gelatinizing basic iron-silicate), also iron phosphates, crenates, ulmates and humates. The ores contain between 20 and 60 per cent. of Fe_2O_3. The phosphoric acid content rises as high as 10 per cent. There is also a mechanical admixture of sand grains and clayey particles."

"Deposits of bog iron ore are found where surface water stagnates in the shallow depressions of flat lands, especially in the vicinity of sluggish streams whose waters are colored brown by dissolved humous acid or humic salts, and in the moor and meadow bottoms of the lowlands of northern Europe, Asia and North America."

2. *Occurrence of Bog Ore.*

Kemp[10] gives the following examples of the occurrence of bog iron-ores.

"The ore-beds [at Three Rivers district in Quebec], furnish ideal illustrations of bog-ore deposits in all their forms. Beginning as a light film, the ore gradually accumulates on the bottom, where it hardens into thick crusts. These are exposed to the sun in the dry season in the shallower reaches, and become very hard cakes. During the succeeding wet season they are again buried under more ore, or sand and ore, until the thickness attained is very considerable. . . . The river flows from the swamp called Grand Plé in the midst of which is a shallow lake called Lac a la Tortue. Ore is dug in the swamp and dredged in the lake. The supply is renewed after being removed."

Beck (Weed)[11] describes the bog ores occurring in lakes in Sweden as follows:

"The lake ores . . . . are found at the bottom of innumerable lakes [in Sweden]. . . . . They are mostly found on a sandy bottom at a distance of about 10 m. from the shore and up to a depth of about 10 m. (32.8 ft.). The deposits are usually thin, rarely reaching 0.5 m. (1.6 ft.) in thickness, but as they may be obtained by simple dredging, they are worked even if but 10 cm. to 15 cm. (4 to 6 in.) thick. The supply is renewed in about fifteen to thirty years. . . . . The ore in the lakes does not form a continuous sheet, but occurs in round or elongated patches, whose direction and arrangement is evidently determined by the currents due to streams entering the lakes, since the ore beds are in shallows covered by an abundant growth of water plants, while the currents supply sand and mud. . . . .

"The formation of these lake ores is accomplished in several stages, each characterized by different material. In the first stage the iron oxide settling on the bottom, at first as a light ochrous mud, gradually hardens into crusts, having the luster, color and hardness of true ore. This mud has a blackish gray, brownish or greenish color, and is filled with vegetal débris. Exposed to the air, it dries to a gray or yellow powder. It is rich in gelatinous silica and contains numerous

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[10] *Ore Deposits of the United States and Canada*, 5th ed., p. 90 (1903).

[11] *The Nature of Ore Deposits*, p. 100 (1905).

algae. On hardening, the masses of mud form either compact lumps (rusor), small or large discs and balls, or else they encrust roots, portions of trunks and branches of plants and animal remains, such as beetles and worm tubes, Phryganid quivers and the like."

3. *Comparison of Chemical Composition of Cuban Brown Ores and Bog Ores.*

a. Condition of the Iron.—The iron of bog ore, as shown by the definitions and descriptions given above, is all limonite (hydrated ferric oxide), while this is not true of the Cuban brown ore.

Analyses show that there is not sufficient water present in most of the ores to combine with the iron oxide to form limonite, especially so since much of the water is combined with the aluminum silicate. There is a considerable portion of the mass highly magnetic, showing the presence of magnetite. The microscopic examination of the ores also shows both magnetite and hematite. The evidence is, therefore, positive that a considerable portion of the iron is in the anhydrous condition as hematite and magnetite. The proportion of these varies in the different samples so that no general analysis can be given. The reddish color of certain samples also shows the presence of hematite. The occurrence of hematite and magnetite in the Cuban brown ores is entirely inconsistent with the definitions of bog ore and therefore of itself would disprove the bog origin of the Cuban ores.

b. Phosphorus and Organic Matter.—The absence of organic matter and the very small amount of phosphorus prove that the ores did not accumulate in bogs where there was much decaying vegetable matter. High phosphorus-content is a characteristic of bog ores, while the Cuban brown ores are remarkably low in that element. In the United States, deposits of bog ores that were once worked have been abandoned, owing to the high percentage of phosphorus.

c. Silica.—As shown in the typical analyses of bog iron-ores given above, silica is also far higher in them than in the Cuban ores. The low silica alone proves the Cuban ores to have had a different origin.

d. Grains of Sand.—The total absence of water-worn grains of sand also disproves the bog origin of the ores. In bogs as extensive as those must have been, were the ore formed in such

places, streams would have entered in many places, and these streams in times of flood would certainly have carried in water-worn grains of sand. Such materials are characteristic of typical bog ores.

e. Chromium and Nickel.—The presence of chromium and nickel in appreciable amounts in the ore is also strong confirmatory evidence of the residual origin of the ore in the decomposition of the underlying rocks which contain these elements. The chemical behavior of compounds of these elements seems to forbid their presence in bog ores except under unusual conditions, and then only in minute amounts, far less than the proportions represented in the Cuban deposits.

Kemp[12] says: "The mineral [chromite] is practically limited to serpentinous rocks and is distributed through them in irregular masses."

On the mine Buena Vista, at Moa, there is an occurrence of Cuban brown ore, which contains 25 per cent. of chromium, or in other words, a type of this ore occurs which is nearly rich enough in chromium to be called chromite.

f. Chalcedony and Quartz.—Finally, the presence of rather large masses of chalcedony and quartz within the iron-ore body cannot be satisfactorily explained by the bog-origin theory. We have seen some of these, several inches in diameter, which could not have been transported by streams entering the swamps without finding them distributed in regular strata and associated with mud and sand. Instead they occur isolated and irregularly distributed throughout the ore-body at Camaguey. Pratt and Lewis[13] describe similar materials in the residuum of the peridotites, where there is no question of their origin. They say that "The weathering of olivine (its decomposition on exposure to the weather) produces hydrous iron sesquioxide (limonite), silica (both quartz and chalcedony), and the carbonates of iron and magnesium. Most of the carbonates are usually carried away in solution."

4. *Occurrence of Cuban Brown Ores.*

a. Deposits Always Found Resting on Serpentine.—The ore-deposits in question rest everywhere upon serpentine or extremely

[12] *Ore Deposits of the United States and Canada,* p. 70 (1903).

[13] *Corundum and the Peridotites of Western North Carolina,* p. 62 (1905).

basic rocks, that have been shown by numerous analyses and microscopic examination to contain iron, chromium, and nickel compounds, sufficient to produce the ores resting upon them by decomposition. Were the ores transported in solution in running water, it would be extremely improbable that the ore would be precipitated only in bogs formed on that kind of rocks. Further, no iron-ore deposits of that character are known in Cuba outside of the serpentine-areas or in close proximity to them, though the serpentines cover only a comparatively small portion of the island.

b. Deposits Occur on Tops of Hills and Plateaus.—The fact that the deposits occur on the tops of the plateaus or hills and on their gentle slopes, and are absent in the lowlands, also disproves their bog origin. Bog ores may occur on flat undrained plateaus, but only when rocks rich in iron are adjacent to the plateaus and occupy higher levels. As yet no one has reported higher-lying iron-bearing rocks that could have been the source of the vast deposits of iron covering the present plateaus of Cuba.

c. Extent and Thickness.—The extent of the deposits, covering many thousands of acres, and the fact that they are continuous, cast much doubt on the origin of the ore in bogs. The descriptions given show that bog ores accumulate in bands near the shores of lakes or ponds, and the deposits are not continuous over extensive areas such as we find in Cuba. Again, bog ores, elsewhere in the world, are thin and must necessarily be so on account of their accumulation in shallow water, where plant-life or humus-materials are abundant. If the deposits are thick, there are alternating strata of iron-ore and sand and mud. The Cuban brown ore is found to consist of solid ore ranging in depth up to 50 ft. Therefore, the extent and thickness of the Cuban brown ores seem inconsistent with their origin in bogs.

d. Physical Character of the Deposits.—Whenever material of any kind accumulates by successive deposition or precipitation of materials held in suspension or in solution, the resulting deposits show lines of stratification or bedding. The complete absence of such evidence disproves the bog theory as an explanation of the origin of the Cuban brown ores.

5. *Summary of Evidence Opposed to the Bog Origin of the Cuban Ores.*

It is, therefore, certain that the Cuban brown ores cannot be regarded as bog ores, since they do not conform to the descriptions of such ores. In chemical composition they differ in the condition of the iron, the amount of phosphorus, the low silica, the absence of grains of sand, the presence of considerable amounts of nickel and chromium, and the presence of masses of chalcedony and quartz. The location of the ores, entirely on serpentine rocks and on the top of hills and plateaus, is not consistent with their origin in bogs. The great extent and thickness of pure ore is unlike bog ore-deposits elsewhere. And lastly, the absence of stratification proves that the ores are not sediments precipitated from suspension or solution.

IV. Proof That the Cuban Brown Ore is not Ocher.

1. *Definition and Description of Ocher.*

The term "ocher" has been used in many different senses, so that there has arisen some confusion regarding its proper meaning. These conflicting views have resulted from the unscientific application of the word to any yellow substance that may be used as paint. The line between limonite iron-ores and ocher has only been drawn arbitrarily, and there is a gradual passage from one to the other. Certain materials are undoubtedly applicable either to the formation of pig-iron or in the manufacture of paints, but in the main the distinctions between the substances of iron-ore and ocher are generally recognized. The definitions of ocher all have reference to the physical character of the material and its chemical composition.

The ocher found in the south of France, called French ocher, has been largely exported for many years to various countries and is favorably known. It is essentially clay rich in limonite, with less than 25 per cent. of iron oxide.

a. The Paint Manufacturers.—Ocher, as defined in the Color Nomenclature Table of the Paint Manufacturers' Association of the United States,[14] is an "important permanent natural yellow color found reinforced with silica, gypsum, alumina, etc. Consists of hydrated ferric silicate of aluminum per-

[14] *First Annual Report of the Scientific Section*, p. 55 (1908).

meating a clay base, and when burnt its shade may be varied." This definition may be regarded as final, as the men most competent to decide what constitutes an ocher are those who make most use of the material.

Sabin.—A. H. Sabin,[15] an authority on paints and paint manufacture, says: "The great supplies of iron-oxide paints are mixtures of these [limonite and hematite], and are found in deposits where the ore is in granular or earthy form, usually mixed with more or less clay; sometimes the clay amounts to two-thirds the weight of the whole, not uncommonly one-half. Such a material is easily reduced to a powder."

Maire.—Frederick Maire[16] says:

> "All ochers are compounds of mixtures of several ingredients or substances. The coloring matter they contain is due to hydrate ferric oxide (limonite) combined with an earthy base, which varies with each locality and sometimes with every hill in the locality where they are found. . . . . There cannot be, therefore, any recognized standard or chemical formula for an article varying as much as this does. They would have to be changed with each new sample that we analyzed. Notwithstanding so many variations, ochers may be grouped into two general classes:
>
> 1. Those where the earth base holding the iron oxide is chiefly of silicate earth.
> 2. The remaining ochers whose base consists principally of clay, earths, or alumina.

b. Geologists.—The following definitions seem to represent the present point of view of geologists generally:

Pirsson.—L. V. Pirsson,[17] Professor of Geology in Yale University, considers ocher as a variety of clay; the clay element being dominant. He says—"When pure it (clay) is white; but it is generally colored red or yellow by iron oxides, forming red and yellow ochers."

"The Mineral Industry."—In the various volumes of *The Mineral Industry*, ocher is classed as a variety of clay. The following statement[18] represents the point of view of the editors. "Yellow ocher is clay which owes its tint to hydrated sesquioxide of iron" (limonite).

U. S. Geological Survey.—Similarly, in the various volumes

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[15] *Technology of Paint and Varnish*, p. 128 (1905).
[16] *Modern Pigments and Their Vehicles*, p. 58 (1908).
[17] *Rocks and Rock Minerals*, p. 328.
[18] *The Mineral Industry*, vol. iv., p. 492 (1895).

of *The Mineral Resources* of the U. S. Geological Survey, ocher has been regarded as bearing a much closer relationship to clay than to the iron-ores.

c. Summary of Definition of Ocher.—Summarizing the above definitions it is seen that standard ochers

1. Must be loose, earthy, and pulverulent in character;
2. Must contain clay (hydrated aluminum silicate) as the base, and it must be dominant;
3. Must contain iron in the form of limonite (hydrated ferric oxide) as the coloring-matter.

2. *Physical Character.*

Authorities agree that ochers are loose, earthy, and pulverulent in character.

3. *Chemical Composition.*

In chemical composition there are wide variations, but there is general consensus of opinion that ochers contain as their essential constituents, clay (hydrated aluminum silicate) as the base, and limonite as the coloring-matter. Some materials high in oxide and low in alumina and combined silica have been classed with the ochers by those who have not been exact in their usage, but there is now a decided tendency among geologists to eliminate from the ochers those materials that are unusually high in iron. Materials carrying more than about 30 per cent. of iron are called iron-ores, and lower are classed as ochers, provided they have the proper physical character and the chemical composition agrees in other respects. Paint manufacturers object to materials low in aluminum silicate and high in limonite. Sabin[19] says that the iron oxides used as paints are mixtures of the iron oxides with clay, and that they are preferable to the heavier pure oxides because "much less liable to rapid settling out of the oil or other vehicle." Thus geologists and paint manufacturers agree that the clay element must be dominant in the ochers, and that the iron present must be in the form of limonite.

4. *Comparison of the Properties of Cuban Brown Ore and Ocher.*

The Cuban ores in question do not agree with the definitions of ocher as given above in several respects.

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[19] *Technology of Paint and Varnish,* p. 131 (1905).

a. Physical Characteristics.—The greater portion of the ores are not loose, earthy, and pulverulent. Of the hundreds of samples seen from the San Felipe district not a single one has the physical character of an ocher. Of the specimens from the Moa district, some do conform to that description, but most do not. It is admitted by all that any conclusions that will hold for Camaguey must also hold for Moa and Mayari, as geologically and chemically the ores are the same, the only difference being a physical one, arising from the different degree of decomposition of the ore in the different localities.

b. Predominance of Clay.—The chemical composition of almost all the analyses, of which hundreds have been made, shows that clay is not the dominant constituent, but instead iron oxide is much more prominent. Therefore, the material does not conform to the standard definitions of ocher. The average of 50 complete analyses of iron-ore from Moa, taken at random from several hundred, and representing all portions of the district and all depths, show 55.09 per cent. of ferric oxide and only 19.9 per cent. of clay possible, if all the silica present is contained in the clay. It is seen, therefore, that the clay is less than one-half the entire material, while the ferric oxide constitutes more than one-half.

c. Condition of the Iron Oxide.—The iron is only partly in the form of limonite, which recognized authorities agree is the necessary condition for the iron of ocher. Practically all the analyses show that some of the iron is in the anhydrous condition as hematite, and the microscopic examination has shown that many samples of ore contain more hematite than limonite, and none were examined in which hematite was entirely absent. That there is much magnetic material in the ore is readily shown by passing an ordinary magnet through a mass of powdered material. An analysis of some of the magnetic material showed that it consisted mainly of hematite, with a small amount of magnetite distributed through the hematite. It is freely admitted that material that can be properly termed ocher may occur in many samples of the ore, but it is so intimately mixed with the hematite, which, according to no definition, can be included under yellow ocher, that it is impossible to separate them. Some experiments to separate the limonite

and hematite in the Cuban ores have been made by washing, but without success.

d. Silica and Alumina.—The analyses run so low in silica and so high in alumina as to prove conclusively that in many of the most ocherous-appearing samples the alumina does not exist as clay, in combination with silica and water, but the alumina is free or merely in combination with water. It, therefore, does not agree with the ochers.

Of hundreds of analyses of ocher that are available, scarcely one can be found that does not contain more silica than alumina, and none were obtained in which the alumina was in excess of the silica by more than a very small amount.

The best Italian ocher has twice, and the best Pennsylvania ocher has three times, as much silica as alumina, while the Cuban brown ores have from one and one-half to 10 times as much alumina as silica. The average of 50 analyses of Moa ores taken at random shows 1.69 times as much alumina as silica. This evidence in itself, showing such marked difference between ochers and the Cuban ores in question, should prevent the latter from being classed as ochers.

e. Economics.—It is admitted that some samples might be obtained from the Cuban brown iron-ore deposits that would consist of ocher alone, but these deposits are unquestionably not large enough to make it possible to exploit the same property for ocher and iron-ore by different concessionaries at the same time. We have examined several deposits of limonite iron-ore and ocher in Pennsylvania where both occur in less intimate association than in Cuba, and yet it has never been found possible to exploit the two together by different companies. The mines in question have been operated for iron-ore at certain times and the ocher separated by washing and thrown away; at other times it has been found more profitable to work the deposit for ocher and in the washing the iron-ore became the waste product. These Pennsylvania deposits represent residual material and are thus similar to the Cuban limonite.

V. True Origin of the Cuban Brown Ore.

The true explanation of the formation of the Cuban ores is unquestionably the segregation or collection of the iron-minerals on the decomposition of the serpentines and peridotites which

originally contained the ore in the form of magnetite, hematite, and as part of the mineral olivine. The microscopic examination of thin sections of the serpentine shows the rock to contain small pellets of hematite thickly and uniformly distributed throughout the entire mass of rock, and some magnetite. As the rock decayed at the surface, the soluble portions of the serpentine were removed, the iron was converted into the hydrated form (limonite) in the main, though some remained as hematite and magnetite. The chromium and nickel, which are common constituents of basic rocks, also remain in the iron-ores. Strangways,[20] in his article, Chrome Iron Mining in Canada, makes the following statement, which is recognized to be true by all geologists: "Chromite has been found only in peridotites and allied magnesium rocks, or in serpentine, which has resulted from the alteration of these rocks."

1. *Evidence of Origin from Serpentine.*

While engaged in the study of the Camaguey ores, six polished specimens were prepared, which plainly showed the gradual transition from rock to ore. All were taken from the San Felipe district, north of Camaguey. No. 1 was a sample of the underlying serpentine, and the large amount of hematite it contained could easily be seen with a lens. No. 6 was a sample of iron-ore from the same locality, analyzing 49 per cent. of iron, while Nos. 2, 3, 4, and 5 were the intermediate phases from rock to ore, arranged in the order named. Here we found indisputable visual evidence of the true origin of the Cuban brown ore.

2. *Description of Formation.*

C. M. Weld,[21] in his paper, The Residual Brown Iron-Ores of Cuba, has given a good description of their formation. He says:

"The ordinary procedure in rock-decay involves the removal of lime, magnesia, and the alkalies, while the aluminous silicates and the ferric oxides for the greater part remain behind. Laterization goes one step further and removes the silica as well. Its characteristics are: (1) the liberation of the silica from its various compounds; (2) the removal by solution of the lime and magnesia; (3) the oxidation of the ferrous to ferric iron; (4) the removal of the silica and the alka-

[20] *Canadian Mining Journal*, vol. xxix., No. 5, pp. 42 to 47 (Mar. 1, 1908).

[21] *Trans.*, xl., 305 (1910).

lies ; (5) the concentration, as a residual mantle, of the alumina and ferric iron, with titania, chromic oxide, and other impurities ; and (6) a sort of secondary dehydration leading to concretionary and pisolitic recemented masses, more or less abundantly disseminated through the mantle.

"With this process in mind, the serpentine may be readily recognized as the parent of the iron-ore. Lime, magnesia, silica, and the alkalies have been largely if not wholly removed, and the iron and alumina have been concentrated. There is seven times as much iron in the ore as in the serpentine, and eight and one-half times as much alumina. About the same ratio appears to hold with the chromium, nickel, and titanium, which are nearly equally persistent with the iron and alumina. In short, there is no need to appeal to a hypothetical foreign source for any of the elements constituting the ore, either in whole or in part. No supposition involving transportation of material is required. Everything is at hand, and the history of the ore, as residual material derived directly from its underlying rock, is complete."

3. *Example of Residual Serpentine Limonite Ore.*

On Staten Island, N. Y., there are deposits of brown iron-ore examined by us that are strikingly similar to the Cuban limonites. The iron-ore occurs in several patches on a serpentine area, and it is there possible to see the gradation from the fresh serpentine through the much-decomposed rock to the iron-ore. In composition and general appearance it would not be possible to distinguish the Staten Island ores from the harder ores of northern Cuba.

VI. Proof that the Cuban Brown Ore is Iron-Ore.

The proper classification of the Cuban limonite ores has been settled by the U. S. government in declaring them to be iron-ores and subject to the duty levied on iron-ores and not that placed on ochers. In the past there have been several cases where attempts have been made to import ochers under the name of iron-ores with the lower importation tariff-rates on the latter, but the decision has been adverse to the importers.

The Pennsylvania Steel Co. and the Maryland Steel Co. have successfully used the Cuban brown ores in the manufacture of steel, so that there can be no question of their value for such purpose.

VII. Classification of the Cuban Brown Ores.

Our general conclusion is that the ores in question occurring at Camaguey and Moa are properly placed under the Third Section in the Classification of Mineral Substances as quoted in "General Bases for the New Mining Legislation," and approved by decree of Dec. 29, 1863.

Exploration of Cuban Iron-Ore Deposits.

BY DWIGHT E. WOODBRIDGE, DULUTH, MINN.

(Wilkes-Barre Meeting, June, 1911.)

DURING April, May, and June, 1910, I was in charge of an examination of the greater part of the Moa iron-ore area in Oriente Province, Cuba, on the north coast, near the east end of the island. My instructions, on arrival at the properties, were to check former estimates of tonnages and grades, and to re-examine the ore comprised in claims covering 44,727 acres. This work included the running of lines dividing the properties into co-ordinate planes, the boring of many thousand feet of holes spaced at the intersections of these co-ordinates, the taking of samples of the ore penetrated, the analysis of these samples for their various constituent minerals, and the determination of the results as to tonnages, depths, and grades, both for individual properties and for the entire group. Each section of every one of the thousand holes drilled was to be compensated for depth and grade with every other, a series of simple arithmetical calculations of no slight magnitude, the mere mechanical labor of which consumed much time, but finally resulted in giving a complete average of all the essential facts for the entire area of 18,000 hectares.

Had it not been for the more than willing, active, and able co-operation of the officers of the Spanish-American Iron Co., from Charles F. Rand, President, and Jennings S. Cox, General Manager, down to the most humble water-carrier, the work would have consumed far more time than it did.

The lands thus systematically explored by me were comprised in the following denouncements: Punta Gorda, Yaminiguey, Baracoa secunda, Sagua, Moa, Yajrumaje, Lirio, and Cabanas, all of which were massed as the Moa group, so called, and cover an area of 13,832 hectares, or 34,179 acres. Some 10,000 acres additional to this was included in neighboring properties, lying between Moa and the east end of the island, the Buena Vista,

Canete, Taco, Barisagua, and Tanamo claims, and a third group of so-called Rodrigo lands, lying in a compact group to the south of and joining the Punta Gorda claim. On all these lands, aside from the Buena Vista and Rodrigo groups, there were found to exist no less than 865,124,000 tons of iron-ore, of an average composition of iron, 43; sulphur, 0.117; phosphorus, 0.012; nickel and cobalt, 0.80; and chromium, 1.7 per cent. In addition to this tonnage there were found some 100,000,000 tons of an average tenor of about 32.5 per cent. of iron. The tonnage found to exist on Buena Vista and Rodrigo was about 250,000,000 tons, averaging 43 per cent. of iron. Considering the fact that all analyses are made dried at 212°, and that the ore carries not far from 14 per cent. of combined moisture, and, say, 25 per cent. of hygroscopic water, this tonnage means about 60 per cent. of the above totals of an iron-ore with iron-content of about 50 per cent.

The preceding papers in this volume, and to which my present paper may be considered an addendum, elucidate, more fully than I could hope to do, the origin and geological character of these ores, some of them with special reference to the attitude of claimants for portions of these ore-fields under the argument that these ores are bog ores or ochers. I will confine myself to the situation, method and expense of exploration, and to probable courses of development and mining, with some attention to the cost of the ore delivered in the United States.

Articles descriptive of the discovery and development of a tonnage of 600,000,000 tons of commercial iron-ore in the Mayari field by the Spanish-American Iron Co., have been published from time to time. Subsequent to these discoveries and their exploitation, the red soil at Moa was recognized as an iron-ore, and researches were immediately instituted to determine the quantity and quality of this ore. These investigations commenced in 1906 and had been carried on almost continuously with a varying degree of vigor up to the time of my own examination, in 1910. The tonnages of this new district proved to be greater than those of Mayari, while the quality was found to be quite similar. The resemblance in grade was but natural, since the origin of ore in these two fields was precisely the same and the breaking-down of the ore-bearing rock has proceeded at Moa in a manner analogous to

that process at Mayari. More than 50,000 acres of land were examined and drilled, the district was mapped, and thousands of drill-samples were analyzed. It was found that the general area of these Moa fields was superposed upon about 60 sq. miles, and that the ore-beds extended directly to the Atlantic shore, forming a blanket more or less continuous from the sea to the summit of the island, the height of land between the Atlantic ocean and the Caribbean sea.

A precipitous range of rugged hills is practically continuous along the north coast of Oriente Province. These hills attain an altitude of from 2,000 to 2,500 ft., and approaching by sea, form the distinguishing feature of the landscape. At points the slopes reach the water's edge, elsewhere they are some miles from the shore. Numerous bays break the coast; some large enough for harbors for ocean-going ships, while others are constricted in area and shallow in depth. A series of coral reefs extending for many miles along the coast protects it from the constant sweep of the Atlantic surge, which is hurled in by the steady NE. trade-winds. Occasionally these reefs are cut by broad and deep entrances, easily distinguishable by the break in the otherwise uninterrupted line of white water that is like a foamy stripe, elongated on either hand until it ends, a mere ribbon upon the blue. These reefs, awash at low tide, are covered at high tide, and so perfect a protection do they form that the decrepit, poorly-rigged, flat-bottomed fishing-boats of the natives are safe inside, no matter how fiercely the combers may smash upon the reefs beyond.

The *ciudadcita* of Baracoa is 35 miles east of Moa, and its history extends back to the time of Columbus, for it was here that he first landed on Cuban soil. The town was founded in 1500. To the west, 50 miles, is the capacious bay of Nipe, where are situated the works and shipping-piers of the Spanish-American Iron Co., the sugar-mills of the United Fruit Co., and a terminus of the Cuba railway. Between Baracoa and Nipe bay there are no settlements worthy the name,—only an occasional fisherman's hovel, where a cocoa-palm grove comes down to the sea, or where there are a few roods of cultivable soil. So much of the scanty earth along this stretch of coast is iron-ore that arable ground is hard to find and is in high request.

Roads scarcely deserve the name in this section of Cuba. While there is the *Camino Real*, the so-called King's Highway, it is impassable for wagons, and from Moa to Baracoa a pack-mule cannot get through, even with an empty saddle. In seasons of high water the roads to Sagua and on to Nipe bay cannot be traversed at all, and communication is almost entirely by boat. The poor transportation increased the difficulty of securing provisions and supplies, of getting and keeping competent men, and of handling the mails.

No surface of soil exists over these ores; indeed, the ore itself is the soil, upon which grow either pine forests or a characteristic tropical jungle. On the lower elevations and in the better drained of the upland interior, pine predominates; inland, where the rain-fall may be heavier, and wherever it remains more permanently after falling, the verdant jungle enters. It closely resembles the jungles of northern South America, with its tough, cord-like creepers, its strange arboreal growths, and its dense poisonous and prickly shrubbery. It is hard to penetrate unless one has in his hands that omnipresent weapon, the *machete*. In the belief that a thin capping of surface-soil and humus might lie above the ore in these jungles, I took a number of samples in these woods at varying depths, which showed on analysis that, when found at all, the ore extended to the surface, whether timbered or not. No stripping of these ore-bodies is necessary to fit them for mining, and during the dry seasons a lighted match may be applied to the forest-floor and the fire will clean off all organic matter above the ore, leaving it free and fit for immediate mining by the steam-shovel or other means of excavation.

Scattered about the surface of these deposits are boulders, flat sheets, pellets, and nodules of hard iron-ore, somewhat dehydrated, and varying from masses of many tons to pieces the size of minute bird-shot. Natives call the pellets *tierras de perdigones*, or "shot soil," a name warranted by their appearance and by the use to which they sometimes have been put, both in peace and in war. While the upper inch or two is occasionally composed entirely of this material, it is usually carried in a matrix of soft ore, and it was the original design, at the time of discovery, to wash this hard ore from the surrounding red soil and ship a product of indurated iron-ore. This scheme,

however, was impracticable, owing to the expense of collecting the hard ore, which is spread over a great area in a comparatively thin layer or appears in isolated deposits and pockets; moreover, the matrix contains so much clay that washing was slow and difficult. During the course of experiments having in view the washing of this material, it was found that the soft-ore matrix was as good ore as the hard, and it was not until this fact was fully realized that the great size and vast importance of these deposits were appreciated and their possibilities realized.

It has been considered by some engineers that these shot ores cemented into masses occur in layers and bedding-planes, and so form a persistent sheet covering a large continuous area. In proof of this they point to the hard boulders frequently found underground in the progress of drilling-operations. Basing my opinion on the results of a drilling-campaign greater than that of any concern aside from the Spanish-American Iron Co., at the Moa and Mayari properties, I cannot agree with this theory. I believe the hard ore found underground in drilling to be blocks and boulders of this cemented material, and not often of large size. Also, that the horizontal outcrops of cemented nodules, at times found along the sides of erosion-cañons, are not original, but have assumed their present condition since they became subject to the changes incident to surface-action; and this is the case whether they are directly upon the top of the ground or near to it. Contrary to statements made in occasional reports, there are in these deposits no definite layers of ore of varying degrees of induration, color, or class. The deposits are homogeneous masses, and the harder ores found so frequently are the result of heat, the action of the elements, and the infiltration of iron salts as a cementing material; while the variations of color and texture are the result of a more or less hydrated condition and a more or less complete disintegration of the original rock, all due to local favoring or retarding causes. I took careful note of the depth reached by nodulized ore and found it to average a few inches, while the extreme depth was 24 ft. This latter depth for nodules was rare; in such cases their proportion of the mass was very slight.

The deposits constitute a surface-mantle varying in thick-

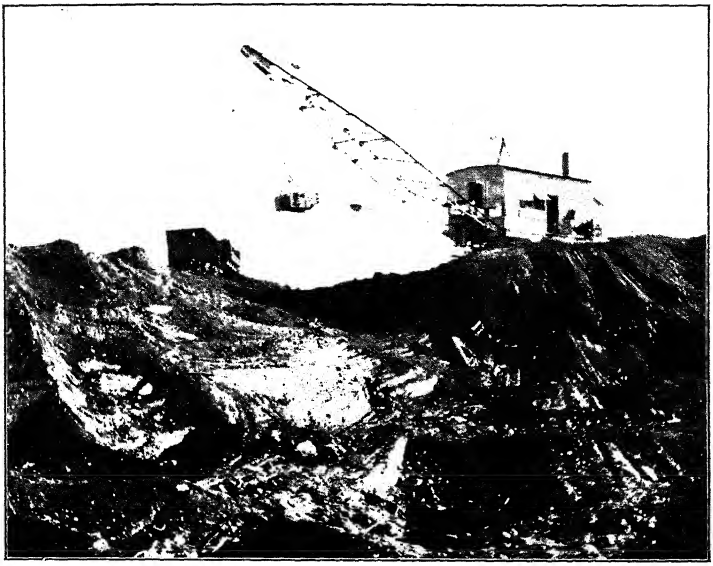

FIG. 1.—A DRAG-LINE EXCAVATOR AT MAYARI, SHOWING RADIUS OF ACTION.

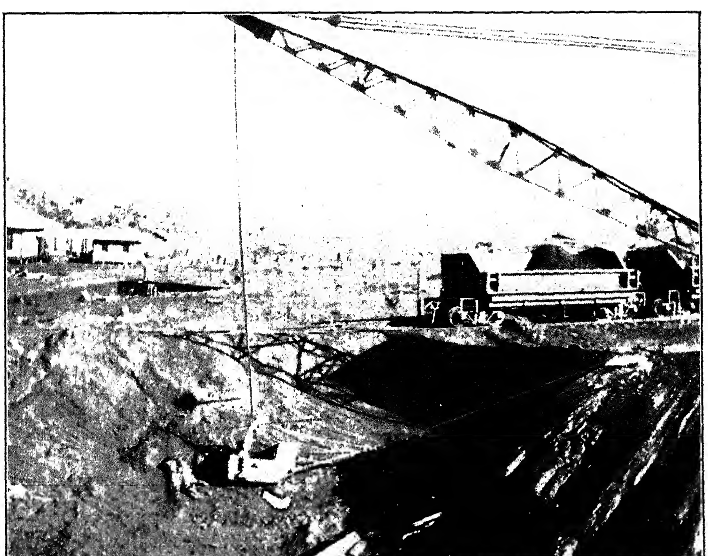

FIG. 2.—THE DRAG-LINE LOADING ORE-CARS AT MAYARI. CARS USED HERE ARE OF 50 TONS CAPACITY AND ARE SIDE-DUMP.

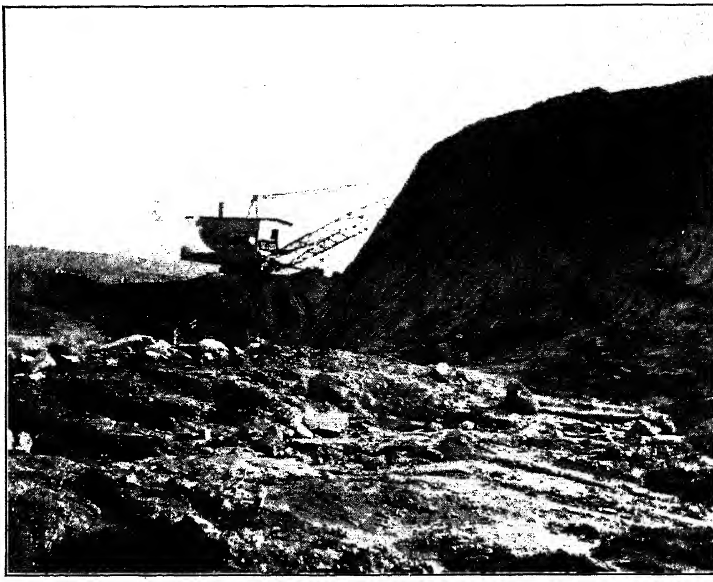

FIG. 3.—ORE EXCAVATED, SHOWING ROCK BOULDERS ON FLOOR.

FIG. 4.—A TRAIL OVER ORE-SOIL IN PINE WOODS AT MOA.

ness from a mere film to 121 ft., which, I believe, is the extreme depth ever drilled in ore in Moa. This hole was bored by men in the employ of the Juragua Iron Co. The greatest depth which I attained was 81 ft., said to be the second deepest ever bored there, and the deepest ever put down by an ordinary crew of two men. There is an average thickness of from 18 to 20 ft.; the results of work under my supervision, covering an area of more than 8,000 hectares of ore drilled, showed an average of 18.83 ft.; Mayari ore, I understand, is a trifle thicker. The thickness of the ore-mantle is affected by local causes, assisting or delaying the breaking-down of the serpentine rock (which experts agree to be the mother of this ore), erosion by streams, and other causes. The ore lies directly upon the serpentine, and mining will be somewhat unfavorably affected by the fact that the gradation from ore to rock is not at all regular, but very rough, so that in cleaning the bottom of an ore-body with any sort of automatic machine, chunks of serpentine are liable to be broken off and lifted with the ore, unless care is constantly exercised. This irregularity is shown plainly at the mines of Mayari, and shipments from these openings to Nipe piers sometimes contain serpentine broken from the floor.

Torrential mountain streams are frequent in this area; a square of 225 hectares was measured for check-work in which were no less than three large rivers with deep gorges, each one worn well into the underlying rock. In this particular area about 25 per cent. of the total was barren of ore. But, while there are many streams, this special case was abnormal and cannot be duplicated in the entire district. In spite of a brief rainy season and a long dry period, waters flow with surprising volume throughout the year. But erosion at the present time is exceedingly slight and entirely negligible so far as tonnage of ore is concerned, for the *arroyo* slopes are hard and smooth, and, even in flood, the rivers bear comparatively little material in suspension.

One peculiarity of this ore is that it stands indefinitely without caving. On exposed vertical faces, open to storm and sun, there is no appreciable sloughing-off of the sides. I have seen pits dug years ago, that have been open to the action of the weather, the vertical walls of which still retain marks of picks

and other tools of the diggers. This ore is very clayey, representative and composite assays showing Al_2O_3, 13.34, and SiO_2, 3.36 per cent. Derived, as it undoubtedly is, from serpentine, the proportion of alumina is naturally very high.

By reason of the character and condition of these ores exploration can be carried on by a process that is simple, accurate, rapid, and cheap. Ordinary 2-in. auger-bits are forged on one end of 4-ft. sectional rods, the other end being fitted to receive a sleeve-nut, 5 or 6 in. long, into which another 4-ft. section may be screwed. As a hole is driven down by the auger-bit additional threaded sections are screwed on the rod, making it any desired length. On each end of each rod, except where the bit is shaped, is a backing-nut screwed down hard, in order to prevent the rods from working too tightly into the sleeve-nuts when turned into the resisting ground, which would render it difficult to release quickly. In most cases ore can be bored by this simple tool with comparative ease, and when hard blocks and boulders are encountered underground, they are sometimes cut by the substitution of a cutting chisel-bit for the auger-point; in other cases the men will move a few feet away and drive another hole, experience having shown that a very short distance will usually be sufficient to avoid a boulder. The hole is started through the drier top soft ore or nodules on the surface, a little water is poured in, the bit lifted and driven down by the combined strength of two men, and then turned in the ore. The work is a combination of churning and boring. Every few feet the tool is lifted, the ore adhering to the bit is cleaned off by pressing a stick into the point of the bit and then revolving the tool, and saved for analysis, and all sludge that has collected above the bit is scraped off. If the hole is sampled in sections, all ore taken out of each section by the bit is saved to make a full sample; but if the hole is sampled as a whole, the ore is all piled upon a cloth and afterwards mixed and quartered down with the ever-ready *machete* to make a suitable sample. When sampling was in sections it was found best to adopt 5-ft. lengths, both for general convenience and to ease the work of the calculator of averages. The drilling is hard work in deep or difficult holes, or where nodules are frequent,—as hard as any labor that a man can comfortably endure. It is done almost entirely by Spaniards,

mostly from the province of Galicia, who become very expert and earn good pay. It is all task-work, and the going rate of contract-wages varies with the depth of holes as well as with the character of country-rock. Each pair of boring-men is accompanied by a water-carrier and a sample-marker, both paid by the day; the sample-marker acts as the representative of the employer. He measures the holes and sees that bottom is reached before the drilling is stopped. The deeper the work the more difficult it is, and there is on the one hand a tendency on the part of drillers to shirk, and on the other to allow themselves extra measurements. They will stop in ore if it is hard drilling, marking *piedra*, or "rock," on their last sample, if there is no one to check them. Were it not for the peculiarity of this ore of standing without caving, this system of drilling would be impossible, and it would be difficult for the engineer to follow and check the depth of holes by dropping down a measuring-rod, or by inserting a bit with which to test the material at the bottom. It is not uncommon to check grades of properties previously drilled by inserting bits in the old holes and reaming out a sample from the sides of the hole. If the original hole has been protected at the surface by plugging it with a piece of sapling, it is very unusual to find the hole caved or destroyed.

The price paid the borers begins at from 1.5 to 2 cents per foot for the first 10 ft. of depth, and increases by the addition of a like sum per foot for each succeeding 10 ft. of progress following. In ordinary ground each borer will earn from $2.50 to $3 per day; in other words, a pair of borers will complete from 10 to 13 holes, averaging 20 ft. deep, per day. Sometimes, when work is unusually difficult, or when it is desirable to get special results on check-work, it is necessary to pay by the day at the rate previously earned on contract, or to give some sort of bonus for depths. Working with one of these drills, two men in my employ drove a hole 81 ft. deep, although it took them two long days to complete it. This hole was drilled at a spot where I thought that the ore was thicker than the original testing, or my own first check, had shown it. The original record was 22 ft. and was marked "rock bottom"; my own check was 20 ft. and was likewise marked "rock." But the third attempt went down four times

as far before it really hit the serpentine, though located less than 10 ft. from either of the others. Evidently both former holes had cut into an ore-boulder that the men thought was bottom, or that they did not desire to penetrate. In the third effort to reach bottom 10 ft. of hard ore was cut by the use of a chisel-bit between the 20- and 30-ft. levels. A fact that was somewhat of a surprise to me, in connection with this hole, was that the bottom section, from 75 to 81 ft., showed ore as high in grade as that in any other part of the boring, and slightly above the average. The borers acquire great facility, and work rapidly and hard. If the ground is easy of entry they complete the holes quickly, and race each other from one location to the next in order to lose as little time as possible. They regard themselves as of a type of laborers higher than the average, and feel pride in their occupation.

In no other way is it possible to explore such an area except at great expense and in a long time. No system of tunnels, pits, or other openings is so well suited to this work. It is well enough to sink pits occasionally, to check by actual observation certain facts that seem patent from the drilling, or to answer questions that may arise. In this manner of drilling there have been bored on that part of the Moa area explored by my men more than 50,000 ft. of openings in ore, counting the work of original explorers and my own check-work.

By this rapid and inexpensive method of boring the ore-blanket it will be possible to determine in advance of any actual mine-operation the precise quality of product to be expected from any given area, and thus to regulate grades won, or to produce any quality within the chemical limits of the ore-body. And it will be a simple matter to ascertain in advance the general topography of the underlying rock, and thus to bring mining-work for years to come under an assured and definite plan and system. All this, of course, means a greatly reduced cost of mining.

To those accustomed to vein-mines or to the great replacement-deposits of the Mesabi iron-range, borings varying from 100 to 300 m. apart may seem utterly inadequate to prove grades and tonnages. But a consideration of the origin of these fields and of their necessarily quite homogeneous character answers this objection in part. The answer is made de-

finite and conclusive and the customary method proved safe by results secured in actual practice. In early examinations of the Mayari field original borings were spaced every 100 ft., but as the work proceeded the ore was found to be so regular in analysis, texture, and thickness that holes were gradually spaced at intervals up to and even exceeding 1,000 ft. There was some variation in the essentials, but the averages proved so closely as to be accepted as perfectly competent evidence. The results reached by these more widely separated borings have been since abundantly proved and confirmed by intermediate holes spaced as close as 50 ft. from each other; while in actual mine-operation over the same ground, shipments also check these distant original holes. My own intermediate lines, run between co-ordinates, were a further proof. Engineers and others accustomed to narrow veins and comparatively small tonnages may be startled at such figures as this work presents, secured, as these have been, on data that may seem absurdly insufficient, but study and examination will convince them of the reasonableness of the assumptions made.

One interesting peculiarity of this ore is that often its appearance is no guide to its analysis. Naturally one might expect the deep reddish soft ore to be of better grade than the coarser, yellowish ore containing grains of quartz, etc. But this lighter yellowish ore, when dried at 212°, is as high in iron as the heavier red-colored ore, and its discovery in a hole is little or no guide to the probable depth of that hole, although it is a fact that this class of ore is found more frequently near the base of the beds than in the higher levels.

It is very important, for this and many other reasons, that any serious attempt at the examination of these ore-fields be assisted by a chemist in the field. About 2,500 samples were analyzed during the course of my work on this examination, most of them in a field-laboratory. It was impossible to maintain an equipment in the hills sufficient for the determination of chromium, nickel, phosphorus and the like, but all iron-assays were made there, and were kept as close to the daily returns from the drillers as was practicable. With the crude equipment at hand, one chemist, assisted by two Spanish grinders from the district, assayed as many as 50 samples in a day. Our laboratory was housed in a palm-thatched hut, one side open

to the breezes, with its foot-thick roof inhabited by snakes, scorpions, and rats, and with myriads of flies, fleas, and gnats swarming about us as we carried on our calculations or weighed out our samples. The NE. trade-winds that come into such a laboratory after sweeping over thousands of miles of sea are freighted with dampness, and it was found that a slight delay in weighing a dried sample caused it to absorb moisture so rapidly as to affect the results. So careful were my selected native assistants in their work of marking samples, both in the field and in the grinding-shed, that of all the samples brought in for analysis less than half a dozen were unmarked or misplaced.

This limonitic ore carries an excessive amount of hygroscopic moisture and is light in weight, varying between 18 and 21 cu. ft. to the ton. At an average of 20 cu. ft., which has been assumed as a safe unit for computation by all explorers in that field, the ore will weigh 5,382 tons per hectare-foot. When the area runs into thousands of hectares, and the average depth to more than 18 ft., it may be seen readily that the estimated tonnage will give an enormous aggregate.

The presence of nickel and chromium has been noted. The former is found in quantities increasing towards the floor of the deposits. In the analyses of several hundred samples for this element, the highest percentage found was 1.28 and the lowest 0.44, with an average not far from 0.80. I need not emphasize the economic importance of an iron-ore averaging 43 per cent. of iron, and carrying 0.80 per cent. of nickel. Several hundred tests for chromium showed an average of 1.75 per cent., a serious matter if it were not that a simple metallurgical process will eliminate this element at one stage of the reducing-operation. These ores are of Bessemer grade, slightly lower in silica than the average Mesabi, and not higher in kaolin than some Mesabi ores. Phosphorus exists in very slight proportion. Sulphur is negligible. At Felton, on Nipe bay, the Spanish-American Iron Co. operates a large works for the beneficiation of this ore by drying it in cylindrical, rotating, horizontal kilns heated to a high degree, which reduces, by 33 per cent., the weight of raw ore charged. Against this cost of nodulization, which may be given at about $1.25 per ton of product of the kilns, are to be placed the saving in freights and

duty, and the advantage to the furnace-man of receiving a partly-prepared material for treatment.

With no over-burden to be removed, the deposit situated close to the sea, with stream-valleys cutting through the ore-beds and running directly to deep water, and with an average thickness suitable for about one shovel-cut, these ores should be mined at low cost by ordinary steam-shovel. The steam-shovel is referred to here as though its advantage for this work were unquestionable, but this is not so certain, since some other type of machine excavator may be better. The drag-line excavator has been tried, and has advantages, especially if the deposit of ore is comparatively thin and the floor quite rough. Also, its radius of action is far greater than that of a steam-shovel, which must be moved frequently. There is no question of the relative efficiency of the two machines if the shovel can get one or two full cuts in clean ore, but such opportunities are comparatively rare. One block of 75,000,000 tons, assaying several percentages better than the average of the district and of a thickness of about 70 ft., can be connected with deep water by a railway 4,000 m. long, without excessive gradients. Ore so situated can be delivered on board ship at an actual operating-cost not to exceed 20 cents per tón. The average cost of mining and rail-transport to the sea for the entire tonnage in sight should be but little more than this amount, if operations are conducted on a scale of magnitude commensurate with the importance of the undertaking.

Iron-ore is transported from Cuba to American Atlantic ports at 85 cents per ton. It is carried in British and Norwegian tramp steel ships of from 3,500 to 6,000 tons cargo-capacity, usually equipped with two cargo-hatches forward and two aft. These vessels do not compare with the great lake-freighters of from 10,000 to 13,000 tons capacity and with from 20 to 33 hatches. To be sure, a lake ore-carrier would not live in the weather to which these boats are subjected, but there is no doubt that reasonably large staunch carriers can be so constructed as to afford rapid loading and unloading at each end of the route. With a ship of this character in this trade, more money could be made at 70 cents per ton than the lake boats make at 85 cents per ton. Adding duties at 75 per cent. of the foreign import rate, incidentals, administration-expense,

nodulizing, and all other charges, a nodulized 54-per cent. Bessemer ore can be delivered from these mines at American Atlantic ports at a cost of about 5 cents per unit of iron, and at Pittsburg at a cost of 8 cents per unit of iron, the additional 3 cents being due to the freight from the seaboard to Pittsburg. The raw ore can be delivered at the same points at 3.5 and 7.2 cents, respectively. Of course Lake Superior ores have a counter and equivalent advantage at Pittsburg as this Cuban product has at ocean ports, due to the cost of the rail-haul between that city and the sea.

Various important steel-making concerns are interested in the Moa region. The U. S. Steel Corporation has a number of men in that field; the Pennsylvania Steel Co. and the Bethlehem Steel Co. are also well represented by their subsidiary companies—the Spanish-American Iron Co. and Juragua Iron Co.; and other Eastern and Western interests are identified with the field. No mining has as yet been started at Moa, but it is probable that operations will not long be delayed.

The Mayari Iron-Mines, Oriente Province, Island of Cuba, as Developed by the Spanish-American Iron Co.

BY JAMES E. LITTLE, STEELTON, PA.

(Wilkes-Barre Meeting, June, 1911.)

Of the several extensive deposits of brown iron-ore in Cuba, including those of Mayari and Moa, that of Mayari was the first to be systematically explored, and was selected as the scene of the first operations in the development of this class of ore.

Construction-work, begun in the spring of 1907, involved the building of 16 miles of standard-gauge railroad and two large double-track inclines, the installation of mining-machinery, a nodulizing-plant, power-plant, shops and shipping-facilities, and the dredging of an extensive basin for deep-draught vessels. Unusual weather-conditions delayed the completion of portions of this work so that the entire plant was not in operation until December, 1909.

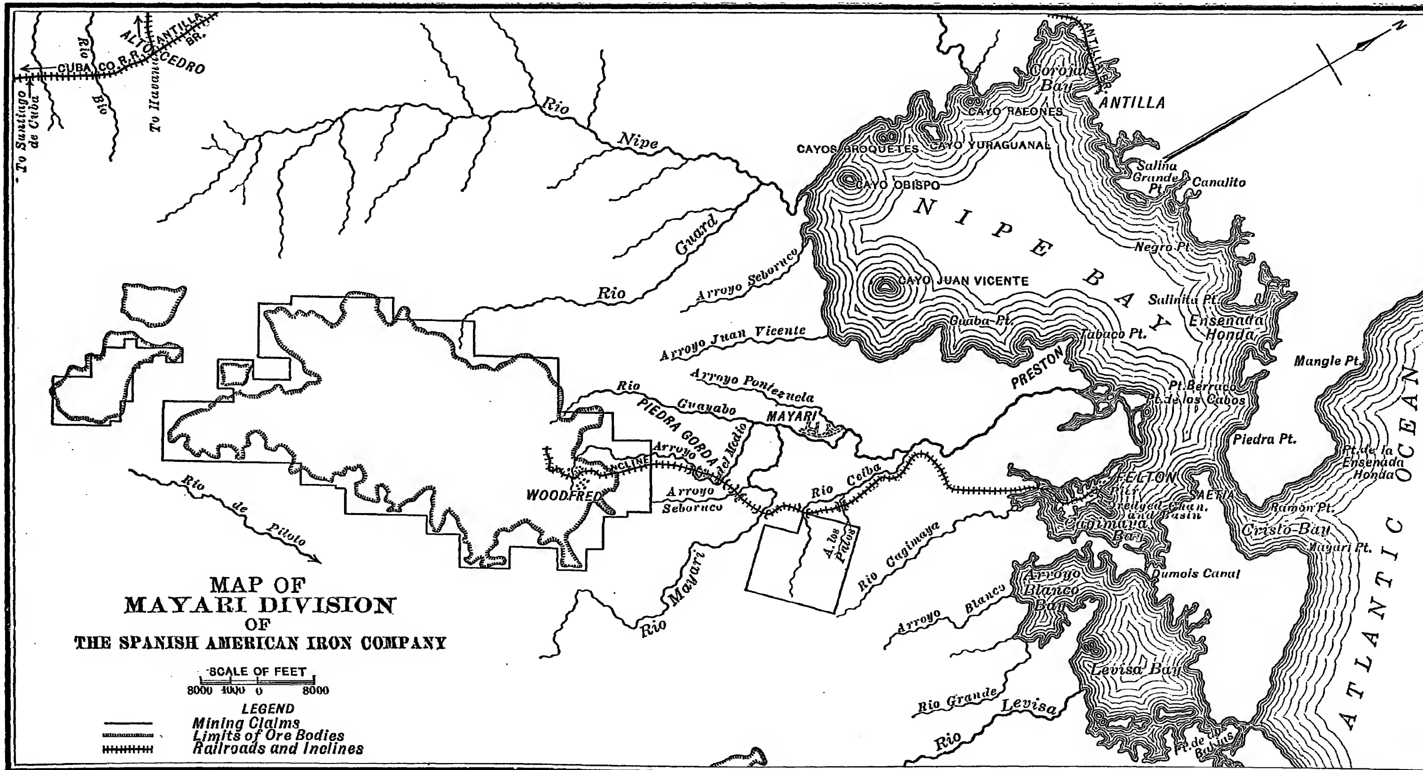

FIG. 1.—MAYARI DIVISION, SPANISH-AMERICAN IRON CO. ORE-BODIES AND MINING-CLAIMS.

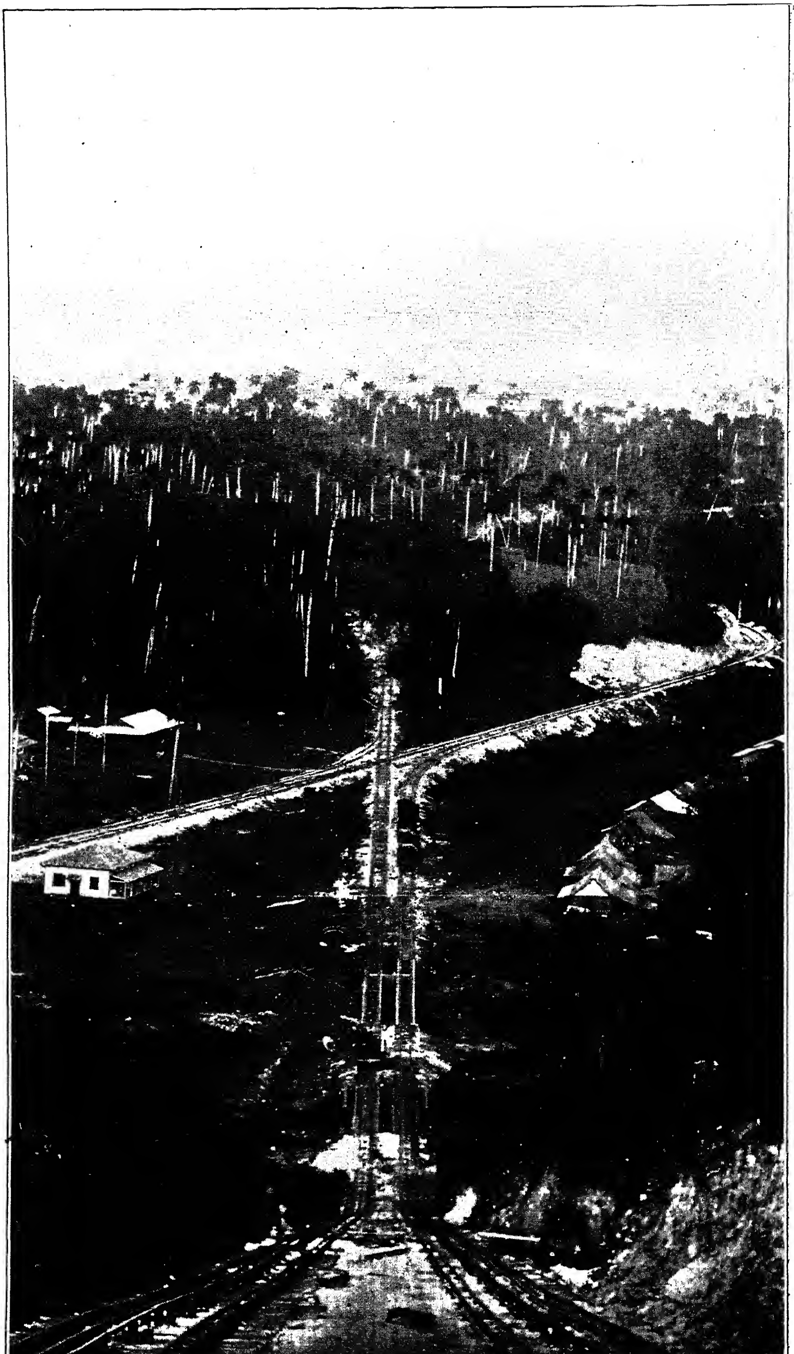

Fig. 2.—View from the Top of the Lower Incline, Showing Piedra Gorda Gravity-Yard.

The ore in its natural state contains a very large percentage of water, which increases to some extent with the depth below the surface. Near the surface it is red in color, with somewhat granular structure. The color gradually changes with depth, finally reaching a bright yellow. The consistency also changes towards the bottom to a clay-like, sticky mass. The relative proportion of red and yellow ore is quite variable; in some places the yellow reaches close to the surface, while in others the red extends almost to the underlying serpentine.

Fig. 1 is a map showing the Mayari division of the Spanish-American Iron Co. The ore lies on an irregular plateau about 15 miles long and 5 miles wide at the widest point, entirely covered with pine trees and brush, which grow directly on the ore. The elevation at the northern extremity, which is approached by the railroad, is about 1,700 ft. above sea-level. At the southern end the general elevation is about 2,000 ft. Ore is removed by means of scraper-bucket excavators and steam-shovels, these machines loading into special standard-guage, side-dump steel cars each of 100,000 lb. capacity. A short haul brings these cars to the head of an inclined plane, 6,800 ft. long, and of varying grades from 6 to 25 per cent. From the foot of this upper incline there is a short railroad about a mile long to the head of a second incline, 1,950 ft. long, and of 25-per cent. inclination. This lower incline ends in a gravity-yard at an elevation of 130 feet.

The ore is therefore lowered by the inclines and connecting railroad through 1,491 ft. vertical height, from elevation 1,621 to elevation 130, the track-distance being 2.44 miles. Both inclines are double track, 14 ft. center to center. Empty cars coming up on one track partly counterbalance loaded cars descending on the other track.

The main cables, 3 in. in diameter, pass over heavy drums, 20 ft. in diameter, at the head of each incline. Each end of the cable is securely fastened to a "barney"-car, against the spring-buffer of which the ore-cars rest. The speed is controlled by a steam-engine geared to the drums, and by post-brakes operated by steam.

From the gravity-yard at Piedra Gorda, at the foot of the lower incline, ore is carried over a single-track standard-gauge line, 13.45 miles long, to Felton, on Nipe bay, where a noduliz-

ing-plant for drying and sintering the ore is situated. The name Felton was selected for the principal town in honor of Edgar C. Felton, the President of the Pennsylvania Steel Co., of which the Spanish-American Iron Co. is a subsidiary corporation.

The nodules produced are stocked under a traveling-bridge at the wharf in position for prompt loading with grab-buckets into vessels for shipment to the United States.

The common labor employed comes principally from the northern provinces of Spain. Many Cuban mechanics and laborers and a few Jamaicans are also employed. There is, of course, quite a colony of American engineers, clerks, mechanics, and others holding positions of responsibility.

The mining settlement, Woodfred, so named in honor of the President of the Maryland Steel Co., is magnificently situated on the ridge of Mayari mountain, overlooking the beautiful Mayari valley, studded with royal-palm groves and tobacco-fields, and, in the distance, the sugar-cane fields of the Nipe Bay Co. along the shores of Nipe bay and Banes bay.

The contour of the ground at the point where excavations were begun, though appearing to be quite regular, is not ideal for steam-shovel operation. The depth of ore is not uniform, in many places the underlying rock projecting far up into the ore, even to the surface. The general slope of the ground, even in the most nearly level places, is quite irregular. Therefore, it is difficult to find many places where it is possible to operate a steam-shovel for an extended period in a cut of economical depth, without including a considerable portion of the rock with the ore excavated. For this reason the scraper-bucket excavators are more satisfactory as well as more economical for excavation, although their capacity is considerably less than that of the large-size shovel used. Three of these excavators are now at work, together with one 90-ton Bucyrus steam-shovel. The excavators operate 1.25-cu. yd. Page buckets, although a larger capacity of bucket is contemplated. The bucket swings through a radius of 60 ft. and without difficulty removes all the ore for a width of about 100 ft. down to the rock bottom, the projecting rock and stumps being discarded. Each machine-crew consists of one operator, one fireman, and three pit-men.

As the machine works up hill or down hill continually, and the track alongside follows the same grade, cars can be dropped down by gravity to be loaded as needed, with a minimum amount of locomotive-service. Three heavy shifter-type locomotives serve the shovel and excavators, and deliver cars to the head of the upper incline. Here the tracks have grades arranged so that cars, when the brakes are released, will run to the incline and the empties coming up will run off on another track. At present two cars are lowered over the incline at once, although, when desired, three cars may be so handled. A brakeman travels up and down with each car to control the brakes at the beginning and end of the trip.

Both inclines are operated on the tail-rope system, the main cable on the upper incline being 3 in. in diameter, of 6 strands of 19 wires each, of plow steel, with a 1⅛-in. independent wire-rope center, the latter also having 6 strands of 19 wires twisted around a hemp core. Its breaking-strength is estimated at 377 tons. The total length of this cable, manufactured by the John A. Roebling's Sons Co., is 7,810 ft.; and its weight exceeded 123,000 lb. The successful manufacture, transportation, and installation of this unusual cable in its present position is a feat of engineering well worthy of notice. Along the track, at frequent intervals, are rollers, 10 in. in diameter, for supporting the cable. These rollers, turned from well-seasoned native hard wood, are carried by a 1.5-in. axle which runs in simple hard-wood bearings spiked to two ties.

At the upper end of the incline the cable passes over two drums, 20 ft. in diameter, set tandem, both carrying heavy gears which mesh with a common pinion, 58.5 in. in diameter, on a center line between the drums. Three half-turns are made over each drum by the cable. The pinion-shaft is also the crank-shaft of a pair of 30- by 30-in. vertical engines which are used in accelerating the moving parts and to carry the cars over certain parts of the incline where the descending loaded cars are on too low a grade to pull the empties up a steeper grade. On each drum are two pairs of post-brakes operated by steam, the load being applied by a weighted lever acting through an eccentric. The surface of the drum is lagged with hard wood for the brake-shoes to act upon. Grooved wooden lagging is also used on these drums for the cable to pass over. The

drums are carried on a shaft 24 in. in diameter in center, with bearings 21 in. in diameter. The construction of the drums and machinery is very massive, steel castings being used for almost all parts. All of this material was furnished by the Nordberg Manufacturing Co., of Milwaukee, Wis.

The machinery-house is 221 ft. back from the head of the incline in order to provide for the cross-over tracks from either incline track to load and to empty track. Spring-switches are used except one, which is free, being thrown by empty cars coming up to the correct position for the next loaded cars going down. Safety-switches are located on the mine-railroad, and at the head of the incline, to prevent damage by cars running away.

The contour of the upper incline, starting from the top, is as follows:

582 lin. ft. on 25-per cent. grade.
2,175 lin. ft. on 17-per cent. grade.
1,917 lin. ft. on 6-per cent. grade.
1,829 lin. ft. on 25-per cent. grade.

At the foot of the last slope the contour ends in a parabolic vertical curve 700 ft. long, connecting the 25-per cent. grade with the 1.4-per cent. grade. An indicator in front of the operator in the machinery-room shows the position of the cars during the trip. In addition there is a complete electric-bell signal-system and an independent telephone-line to provide for communication between the top and bottom of the inclines. The main rails on incline are 100 lb. per yard. Between these is a barney-car track of 36-in. gauge, using 56 lb. per yard rails.

The connecting railroad from the foot of the upper incline to the top of the lower line is arranged with suitable cross-over tracks so as to facilitate handling the cars up and down by one locomotive, which is of the same type as those used on the mine-railroad. Gravity-tracks at both ends provide for handling cars to and from the inclines.

The lower incline, arranged exactly the same as the one above, is 1,950 ft. long, with a uniform 25-per cent. grade ending at the bottom in a long vertical curve. The cable used is of the same diameter as the upper incline cable, in order to keep the machinery details uniform, but, as the length and

weight of cable are much less, it is made of cast steel instead of crucible steel, and has hemp center instead of an independent wire rope.

At the foot of the lower incline is the Piedra Gorda gravity-yard, where loads are made up into a train by gravity, and empties, taken from train by a switch-back arrangement, are run to the foot of the incline. Fig. 2 is a view from the top of the lower incline, showing the Piedra Gorda gravity-yard. The main-line railroad leads off to the right, towards Felton. The Mayari valley and Nipe bay are shown in the background.

The main-line railroad from Piedra Gorda yard to Felton, first-class in every respect, is single track, 90-lb. A. S. C. E. standard rails, laid on native hard-wood ties, ballasted with rock. The locomotive which handles the trains on the main line is of the consolidation type, with leading truck and trailer-wheels, cylinders 19 by 24 in., capable of hauling a 45-car train of ore, weighing about 3,200 net tons, from Piedra Gorda to Felton. The maximum grade is 0.5 per cent. in favor of loads, and the maximum curve is 6°. All bridges are of steel.

At Felton, the terminal on Nipe bay, the trains, after weighing the cars, are delivered to a track on the west side of the raw-ore yard, where one side of a car rests on a sill wall. A plan of the works at Felton is given in Fig. 3, and Fig. 4 is a section through the raw-ore yard and the feed-end of kilns, looking north. The full length of the yard is 750 ft. Two electric gantries carrying trolleys with 6-ton grab-buckets cover this distance and handle the ore from the yard to the kiln-feeders.

The gantries also carry machinery for dumping the ore-cars. A pair of hooks, suspended from hoisting-drums, are guided by means of "tag lines" so as to engage with pins in the door on the front side of the car. In raising, the door turns about hinge-pins situated on the back side of the car so that when the door is raised to its full height, through about 90°, its frame is in position to serve as a link in lifting the back side of the car-body off of the under-frame through a sufficient angle to discharge its load over the sill-wall on which the front of the car-body rests. As a rule, cars will dump when raised to 45°; or perhaps a little higher, if the ore is particularly wet and sticky.

The nodulizing-plant, located on the east side of the raw-ore yard, consists of 12 rotary kilns, 10 ft. in diameter, and 125 ft. long, set at an inclination of $\frac{5}{8}$ in. per foot, and 20 ft. apart. The kilns are of the type commonly used in the manufacture of cement. The diameter, however, is unusually large in order to

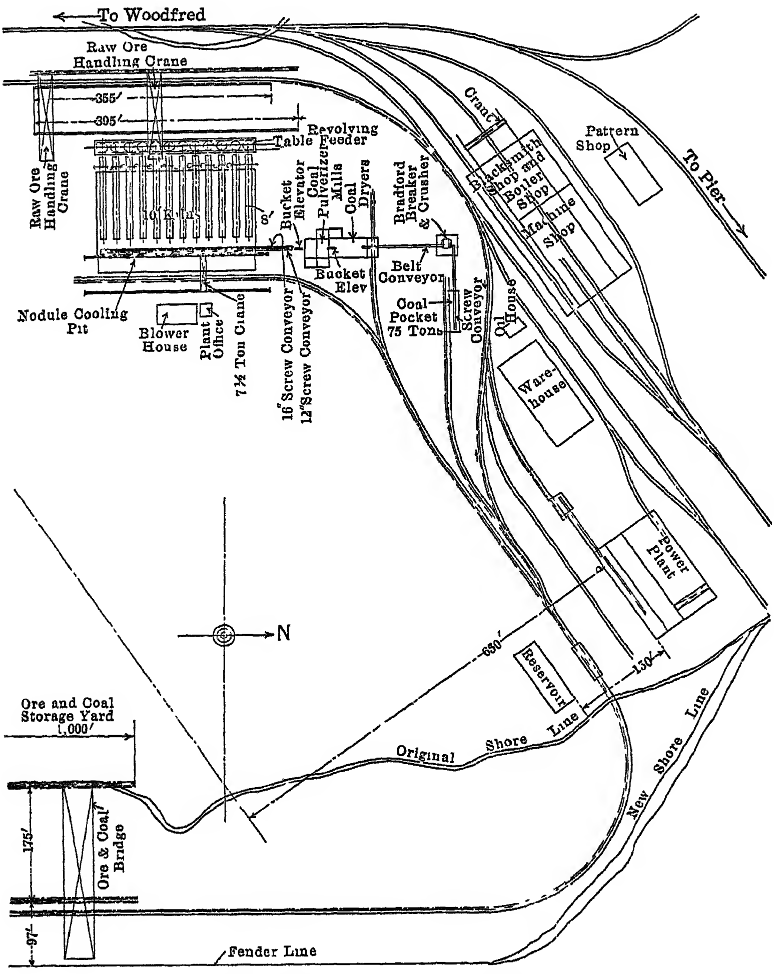

Fig. 3.—Plan of Works at Felton.

overcome trouble from "ringing-up" in the hot zone, which often causes serious delays in the operation of kilns of smaller diameter. Each kiln is carried by two steel tires rigidly fastened to the shell. The cut-steel driving-gear attached to

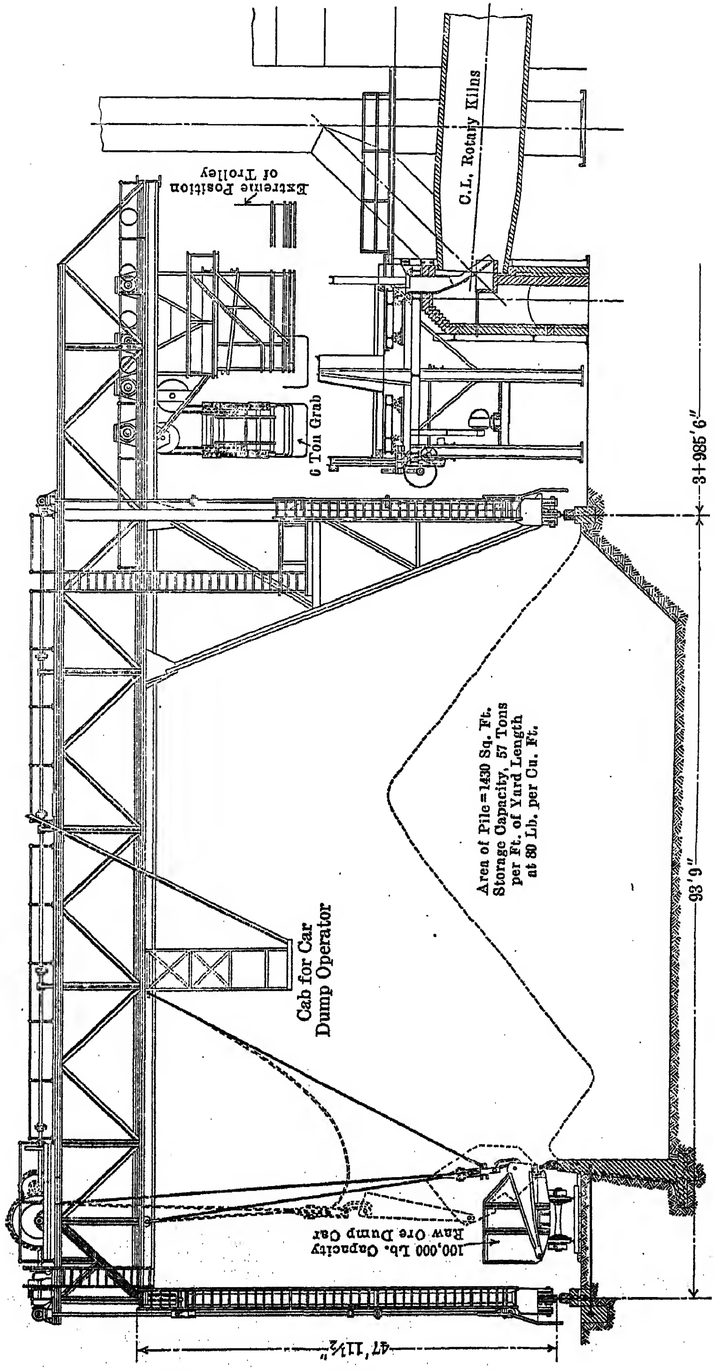

FIG. 4.—SECTION THROUGH RAW-ORE YARD AND FEED-END OF KILNS (LOOKING SOUTH).

the shell close to the tire near the cold end is 152.78 in. in diameter, and 4 in. in pitch. Each kiln is driven by a 35-h-p. variable-speed motor.

To protect the kiln-drive motors from heat and dirt, they are placed in a tunnel of reinforced concrete, running from end to end of the kiln-building. Belts from the motors pass through slots in the side-wall of the tunnel to the pulley on the driving-gear. A fan at each end of the tunnel supplies fresh air.

The kilns are lined for 85 ft. from the hot end with 9-in. fire-brick; the remaining 40 ft. has a 6-in lining. At the upper end each kiln opens into a brick hood connected to the stack, 78 in. in diameter, and 90 ft. high above the center of the kiln.

The device for feeding the raw ore into kilns is very simple and effective. A table 19.5 ft. in diameter is revolved at a speed of about one revolution per hour. One side is kept filled with ore by a grab-bucket on the raw-ore yard gantry, the cantilever extension of which covers the feeder-table. Under the edge of the kiln side of the feeder-table is a wide hopper ending in a chute set at a steep angle. The ore is gradually and regularly plowed off the table by a fixed deflector, which makes an acute angle with the direction of motion of the ore on the table. The ore falling through the chute is delivered to the kiln several feet from the end.

The sintered ore in the form of nodules falls from the delivery end of the kiln to an open cast-iron chute set at an angle slightly steeper than 30°. This chute passes under the floor, delivering the nodules outside of the building into a trough 12 ft. wide and 9 ft. deep below the end of the chute. This trough extends the full length of the kiln-building, 240 ft. A small stream of water runs down each trough with the nodules, facilitating the motion and furnishing the water for cooling. A pump is provided to remove any excess of water in order to maintain a depth of 8 or 9 ft. in the trough. A 7.5-ton overhead electric traveling-crane, carrying a man-trolley with 3-cu. yd. grab-bucket, is provided for removing the nodules from the trough and loading them into 50-ton electric transfer-cars on the track passing alongside of the trough.

On the north side of the kiln-building the coal-pulverizing plant is located. Coal is brought from the wharf in the same

transfer-cars as are used for removing nodules. It first passes through a Bradford breaker and a roll-crusher, which break it down to 0.75-in. size and less, at the same time removing the foreign materials. Crushed coal is taken from the crusher by an 18-in. belt-conveyor to the 150-ton storage-bin. From the bottom of this bin the coal runs into two rotary driers 48 in. in diameter, 30 ft. long, in which it is dried to 0.5 per cent. or less of moisture, in order to be in condition for pulverizing. The dried coal is elevated by a bucket-elevator to a small bin, which feeds to four 42-in. Fuller-Lehigh pulverizers placed in two pairs on either side of the screw-conveyors into which the pulverized coal falls from the bottom of the pulverizers. Motors for driving the Fuller mills, driers, elevators, and conveyors are placed in dust-proof brick buildings on either side of the main building; the shafting extending through the walls carries the driving-pulleys.

The pulverized coal from the screw-conveyors under the mills is taken by bucket-elevators to a 16-in. screw-conveyor about 300 ft. long, with an opening in the bottom for supplying the coal to small bins opposite each kiln. The bottom of each bin forms a hopper for a short screw which feeds the coal regularly to the low-pressure burners.

The blast, 9 oz. pressure at the nozzle of the fan, is supplied by four Buffalo blowers, situated in a separate building to the east of the kiln-building. All blowers deliver into a common pipe, which passes over the nodule-crane runway, down into the kiln-building, and to the various kiln-burners.

At the wharf is a stock-yard 1,000 ft. long, covered by two electrically-operated ore-bridges, one carrying a 15-ton trolley and grab for handling ore; the second a 6-ton grab to be used principally for unloading coal and also as an auxiliary ore-bridge. Both bridges have a main span of 175 ft. and a cantilever extension on the water-side 90 ft. long, to the end of which is hinged an additional 60 ft. to carry the trolley out over the hatches of vessels in loading ore or unloading coal. The latter extension or boom is arranged to be lifted to clear the ships' upper works in moving from hatch to hatch. Fig. 5 is a section of the ore- and coal-storage yard, looking south.

The construction at the water-front is somewhat unusual. Close to the front leg of the bridge, and parallel to its runway,

is a trestle extending over one side of a trough. A transfer-car brings the nodules from the nodulizing-plant, and discharges from one side into this trough in position to be readily loaded into the vessel, or to be moved back to storage under the main span of the bridge by the grab-buckets. The bottom of the trough is 1 ft. above high tide. Its outer wall is formed by planking spiked to a row of piles. All of this construction, being above the water-line, is not subject to damage by the *teredo navalis.* From the outside of the trough-wall the bottom drops off at an angle of 45° to 28 ft. deep at the fender-line, which is approximately under the hinge of the boom of the bridge.

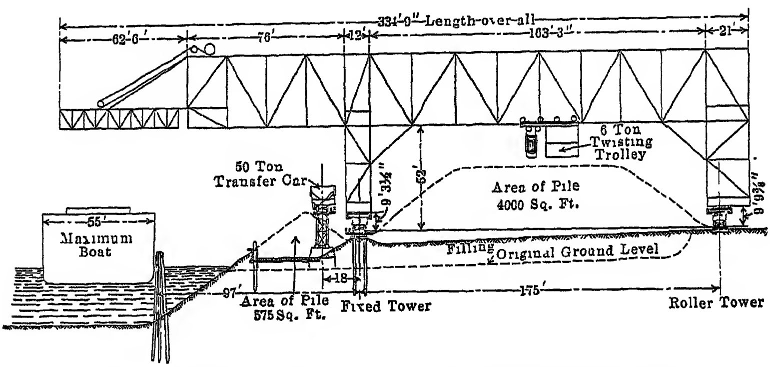

FIG. 5.—SECTION THROUGH ORE- AND COAL-STORAGE YARD (LOOKING SOUTH).

Considerable dredging was necessary in order to provide a suitable harbor. A basin 1,500 ft. long, 200 ft. wide at each end, and 400 ft. wide at the widest point, was dredged to a depth of 28 ft. The approach-channel, 2,500 ft. long and 200 ft. wide, was dredged to the same depth. Felton, on Cagimaya bay, a well-protected branch on the south side of Nipe bay, close to its entrance, has proved a very safe and satisfactory harbor.

As a part of the Felton plant a large repair-shop has been installed, with ample machine-tool equipment, foundry, blacksmith-shop and car- and boiler-repair equipment. The main floor and the yard on the west end are covered by a 30-ton overhead electric traveling-crane. All except the smallest machine-tools are individually driven by electric motors. A

well-arranged lavatory and toilet-room is provided for the comfort of the employees. The building covers a space 120 by 260 ft. In a separate building is a carpenter- and pattern-shop, 40 by 90 ft., containing a saw-mill. Near the machine-shop is a general warehouse, of steel, 70 by 150 ft., included in which is a bonded warehouse for the use of officials of the Cuban Custom House.

Nearby is the electric power-plant, in which are installed three 500-kw., 250-volt, D. C. Crocker-Wheeler generators. Steam is furnished by two batteries of Babcock & Wilcox boilers, each of 880 boiler h-p. capacity, operating at 150 lb. pressure. The engines are cross-compound Wisconsin Corliss, with cylinders 20 by 40 by 42 in. Weiss barometric condensers, with auxiliary pumps, complete the equipment. Coal is delivered to a bin at the west end of the building by the electric transfer-car; from this bin it is taken by conveyor to a crusher, and subsequently elevated and conveyed to an overhead suspended bunker, under which is a chute for feeding the hoppers of the stokers. Ashes are handled by an elevator and delivered to a car on a track alongside of building. All the plant buildings are of substantial steel construction covered with corrugated iron, and in most cases having a concrete floor.

To provide accommodation for employees, towns have been established at Felton and at Woodfred. The Felton establishment includes two hotels, three *fondas* or eating-houses, a steam-laundry, bakery, general store, butcher-shop, ice-plant, and many dwelling-houses of different grades. The general water-supply comes from the Mayari river, which is crossed by main-line railroad, about 11.5 miles from Felton. At this point water is pumped to a tank on the top of a nearby hill, from which it flows by gravity through an 8-in. pipe-line to Felton.

At Woodfred a well-equipped hospital has been established under the care of a competent physician and surgeon. Its situation, on top of Mayari mountain, is ideal for the purpose. The branch hospital at Felton cares for accident cases and is used also as a dispensary. Sanitary conditions, both at Felton and at Woodfred, are carefully guarded, so that the percentage of sickness among employees is very low.

In addition to the wharf for the receipt of coal and the ship-

ment of ore, there is a pier constructed of creosoted pine, 1,700 ft. long, to a point where 25-ft. depth of water is reached. This pier is used for the receipt of general merchandise and machinery, and for local passengers.

Transportation of ore and coal is handled at present by chartered steamers, ore being delivered to the Maryland Steel Co. at Sparrow's Point, Md.

Nipe bay is a growing sea-port, with weekly communication with New York by the Royal Mail Steam Packet Steamers. It is also served by the bi-weekly service of the Munson line. It lies on the north coast of Cuba, almost directly north of Santiago de Cuba. The NE. trade-winds, which blow perhaps from 60 to 80 per cent. of the time, moderate the temperature and make this part of Cuba quite a desirable location for an American colony.

The Spanish-American Iron Co. is also operating the hard-ore mines of the Daiquiri group, on the south coast of Cuba, about 15 miles east of Santiago. The main ore-property at Daiquiri, once considered as three separate mines, San Antonio, Lola, and Magdalena, has now developed into a practically continuous body of ore. Fig. 6 is a view of the Lola mine. In this view the ore can easily be distinguished from the waste by its darker color. Fig. 7 is a side-view of Lola hill, showing the San Antonio mine in the center, and the Lola stripping higher up. The waste-banks are on the right and the ore-lowering inclines on the left. Both the ore and the over-burden are removed from a series of benches. Fourteen steam-shovels are employed for stripping, the largest of which is a 90-ton Marion carrying a 4-yd. dipper. All are served by locomotives and trains of side-dump cars for removing the rock to waste-banks on the back side of the hill.

On account of rock being mixed more or less with the ore, it is necessary to load all of the ore by hand into small cars, which are run to lowering-inclines. These inclines carry the ore in skip-cars to the main-line railroad, which runs from the foot of Lola hill to La Playa, the shipping-port at the coast, 4 miles from the mines.

A hoisting-incline is provided for raising coal, machinery, and general supplies from the main-line railroad to any level of

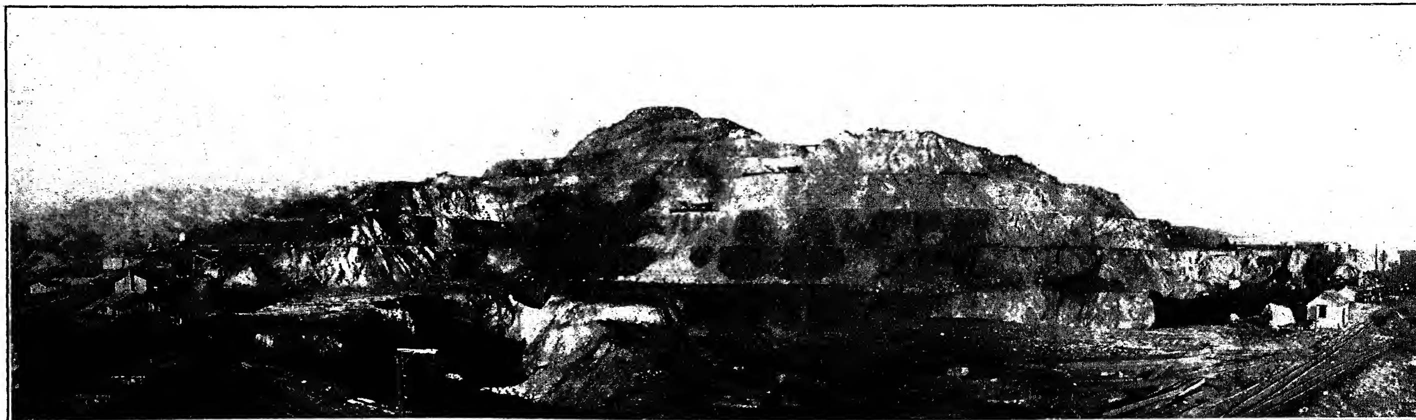

FIG. 6.—LOLA MINE, DAIQUIRI GROUP.

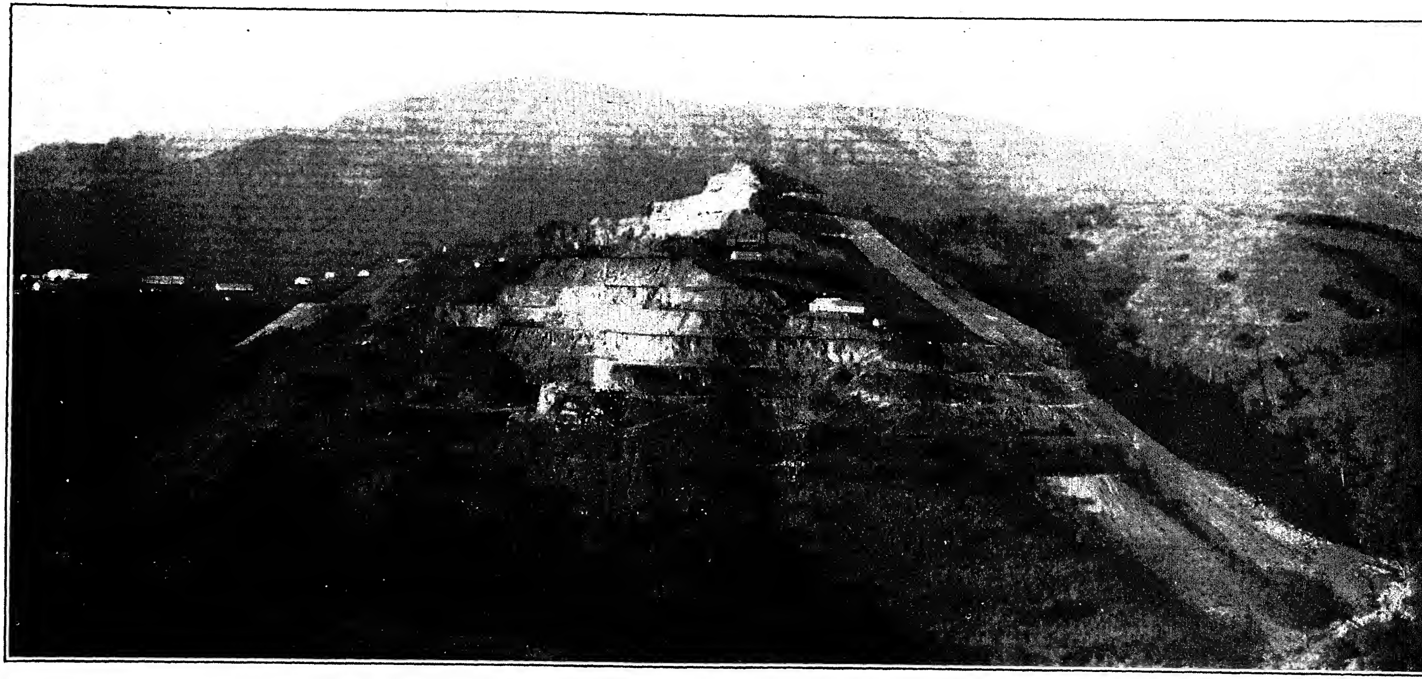

FIG. 7.—SIDE-VIEW OF LOLA HILL, DAIQUIRI.

the mine. A modern air-compressor plant is situated along the railroad near the San Antonio mine, and a pipe-system is arranged to furnish compressed air for tunnel-exploration and for general service to any part of the mine. Steam-drills are used in principal blasting-work.

Ore is also brought from the Berraco and Sigua groups of mines, located to the east of Daiquiri, over a narrow-gauge railroad joining the standard-gauge main-line about 2 miles below Daiquiri mines. Before shipment, all of the ore is crushed in a Gates crusher-plant to sizes suitable for use in the blast-furnaces.

Exploration-work is carried on very systematically and carefully at Daiquiri. That this has been successful is shown by the fact that for years the ore in sight by very conservative estimate at the end of each year is more than at the beginning, although half a million tons or more of ore are mined annually.

The Daiquiri and the Mayari mining-operations are carried on by independent organizations, each plant having a complete repair-department, laboratory, and office force, all under the general supervision of Charles F. Rand, President, and Jennings S. Cox, Jr., General Manager. General offices are maintained in Philadelphia, Pa., and Santiago de Cuba.

The Preparation of Brown Iron-Ores.

BY H. S. GEISMER,* BIRMINGHAM, ALA.

(Wilkes-Barre Meeting, June, 1911.)

Introduction.

There are three general methods available for obtaining commercial brown iron-ore: hand-screening; washing; and washing and concentrating.

Hand-screening has produced a large tonnage of ore in the past, but is rapidly falling into disuse, except as a preliminary step, largely because modern furnace-practice requires all foreign material to be separated from ores before they are deliv-

* Keiser-Geismer Engineering Co.

ered to the furnace, and hand-screened ores can rarely be made to fulfill that requirement. Another argument against hand-screening is, that the waste-pile often contains as much ore as the screen recovers. The operation known as hand-screening consists of throwing the ore-bearing material against a stationary inclined screen of 0.5- to 0.75-in. mesh, similar in construction to the screens employed in sand-pits.

The recently-prospected brown-ore deposits of Cuba seem to offer possibilities for hand-screening not possessed by the average American deposit, since these ores are not associated with worthless clays.

Washing, and washing and concentrating, may be discussed together, since concentrating is never practical on brown iron-ore, except to treat ores that have already been washed. (Strictly speaking, washing might be classed as concentrating, but it is never so designated by brown-ore men.)

The various steps of removing ore-bearing material and converting it into ore suitable for blast-furnace use, may be grouped under five heads: loading for transportation; transporting; feeding into the washer; washing; and concentrating. These five subjects will be considered in consecutive order.

Each of these successive steps may be accomplished in a number of different ways, the choice depending entirely upon the character of the ore-bearing material to be treated.

Brown-ore deposits vary in character:

a. As to size of the ore-fragments—which may vary anywhere from gravel to boulders weighing tons.

b. As to material with which the ore is associated—clay, sand, loam, gravel, and chert are all to be classed as ore-bearing material.

c. As to richness of deposits—one yard of ore-bearing material may yield one ton of ore, while another yard may yield but a thimbleful.

d. As to thickness of deposit—anywhere from blanket deposits a few feet in thickness to concentrate deposits several hundred feet thick.

e. As to quality of ore—anything from pure limonite to ferruginous sandstone.

f. As to origin—you may choose between "residual," "replacement," and "concentrate;" no matter which of these terms

seems to fit your particular deposit, there will be plenty of geologists ready to convince you that your conclusions are erroneous—and perhaps they are. But this question need not be considered in the present inquiry.

I. Loading.

All the methods for loading and transporting earth and rock are applicable at times to the different classes of brown-ore deposits. Those most used are: loading with pick and shovel into wheel-barrows, dump-carts, and wagons drawn by mules, horses, or oxen, or tram-cars, drawn by animals or dinkeys; loading with steam-shovels into drop-bottom wagons, tram-cars, or side-dump cars of various styles and capacities, from 1 to 12 cubic yards.

The greatest difference between the loading of brown-ore dirt and that of earth for railroad-building or similar purposes, is that the brown-ore material cannot always be loaded blindly, but requires careful handling to prevent mixing the ore-bearing material and the worthless rock with which it is often found intimately associated, and which cannot always be separated from the ore by washing. This consideration often eliminates the steam-shovel as a loading-device in ore-banks that would be otherwise well adapted to that style of loading; and instances are not rare in which washers fed by steam-shovels have been abandoned on account of the low quality of ore produced, and at a later date have been successfully operated under management that substituted hand-loading for shovel-work.

Brown-ore deposits are often buried under blankets of worthless clay and sand that require to be stripped off before the ore can be recovered. For this class of "stripping" the steam-shovel is admirably adapted, and the foregoing remarks concerning the care needed to separate ore from foreign material when loading with shovel do not here apply, since the line of separation between the ore-bearing material and the overburden is well defined. It is generally advisable to carry on such stripping in advance of the mining. The various types of "drag-line excavators" now manufactured present great possibilities in connection with stripping-operations, especially where large lean and barren areas have been left surrounded by deep

cuts, and subsequent operations require the wasting of these abandoned areas, to permit of mining the ore which lies at deeper elevations.

The steam-shovel is usually a "losing proposition," if used in mining shallow deposits. Cuts under 5 ft. in depth can be handled more economically by hand, unless the deposit consists of ore-masses too large for hand-loading. This mention of large masses suggests another manner in which the steam-shovel may interfere with the output of a washer. I refer to the tendency to load with the shovel-dipper masses larger than the crusher at the washer will handle, necessitating extra labor and costly delays. The economical place to break down large lumps is in the ore-bank, where dynamite can be used freely; but the irresistible tendency in steam-shovel work is to load anything that can be handled by the dipper. It would, of course, be possible to install crushers capable of receiving any boulder that could be loaded into a shovel-dipper; but, except in rare instances, the cost of such crushers would be prohibitive for brown-ore plants.

In districts where steam-shovels are used to advantage, great variations are noticeable in the type of shovels employed. This generally indicates that at various times the financial standing of the different operators has varied; and the date on the name-plate of the largest shovel will probably coincide with the date of the best-filled treasury. The tendency in brown-ore mining is towards larger shovels with standard-gauge railroad-trucks. The old argument that widely-separated deposits at varying elevations require light-traction shovels that can be more easily moved about, is now answered with, "small, widely-separated deposits do not require steam-shovels at all."

The light revolving type of steam-shovel was, for a time, very popular in brown-ore work; but, except for following narrow leads of ore, it cannot compete with the large standard type of equipment. The operating-costs are nearly the same with the large and small shovels, while the outputs are almost as two to one.

II. Transporting.

The system of transportation to be adopted is regulated largely by the method employed for loading the ore, but each

system may permit of variations that may involve startling results. For example, a deposit made up of small scattered pockets, lying at widely-different elevations, can only be loaded by hand-labor. This would suggest transportation by wagons or mule-carts; but, owing to the depth of some of the pockets, the grades become too steep for mule- or horse-travel; oxen are substituted, and no further trouble is experienced. Again, a change in car-design, where self-dumping cars are employed, may greatly affect the economy of transportation.

The richness of a deposit needs small consideration in planning the method for loading, but it has a marked effect on the cost of transportation. The cost of transportation per ton of ore produced will depend upon two conditions: the richness of ore-bearing material (number of cubic yards of material required to produce one ton of ore); and the length of haul from ore-bank to washer. The first is dependent on natural conditions. If the deposit is so lean that transportation by any known system is economically impracticable, there is nothing left to do but abandon the property. The second is largely dependent upon individual judgment in choosing the location for the washer. Present practice seems to favor a large central washing-plant, fed by a number of scattered ore-banks, in contrast to a small washer, located at each deposit. This practice lowers the actual cost of washing the material, but introduces an excessive transportation-charge that very often more than offsets such saving.

The thickness of a deposit may limit the choice of systems of transportation that can be adopted. For example, a thick deposit of small area can only be followed down by means of a hoisting-engine in connection with an incline or shaft.

The method adopted for loading the ore-bearing material may determine the system of transportation required. Mechanical loading, for example, requires mechanical haulage.

Where dinkeys and side-dump cars are used for transportation in connection with shovel-loading, it is generally advisable to have this equipment and the shovel of standard-gauge design; this allows great flexibility in the shifting of the equipment, besides making it possible to handle railroad-equipment, such as car-loads of coal, to any point of the operation.

In connection with shovel-loading, the transportation requires

careful watching. Unless the shovel can be kept at work continually it will not show a satisfactory cost; and the only way to accomplish this is so to arrange the system of transportation that the shovel-loading track is always supplied with empty cars.

III. Feeding the Material into the Washer.

When the ore-bearing material arrives at the washer it is dumped on to a grizzly, made up of parallel iron bars or rails with 3-in. openings between them. The fine material which falls through the openings passes directly into the log-box or falls into a mud-box and is carried to the log-box by a flume. The coarse material rejected by the grizzly may be handled in several different ways: broken down with hammers, as it rests on the grizzly-bars, until it will pass through the openings; or carried down the grizzly-bars (this requires that the bars be set at an inclination of about 30°), and delivered into a crusher, from which it passes into the log-box; or delivered from the end of the grizzly-bars directly into an overhead screen, the fines passing through the screen into the log-box while the coarse goes directly into the loading-bin.

The output that can be obtained from a washer is often limited directly by the amount of material that can be handled through the grizzly; and yet, in most instances, little attention is paid to the grizzly design, either as to the area required for it or the manner of delivering material to and from it.

To design a grizzly properly, two determinations are required: (*a*) the size of the ore-fragments as delivered to the grizzly; and (*b*) the character of ore-bearing material (mud, clay, sand, gravel, etc.) delivered upon the grizzly with the ore.

When the ore occurs principally as large "dornicks," the grizzly-area must be large. If the grizzly-bars are horizontal, a large number of men will be required to hammer the lumps through, and if they are inclined, the material must be spread over a large area, to prevent the lumps from blocking the openings. A nozzle, delivering water at about 100 lb. pressure, is of great service in separating the fine material from the coarse and forcing the fines through the bars.

If the ore occurs in small pieces, varying in size from gravel to cobblestones, imbedded in stiff clay, the grizzly must be

designed to effect a partial breaking-down of the clay mass, or the resulting product at the washer will consist largely of clay balls. This can best be accomplished by the use of giant nozzles. However, it is not always possible to separate the ore sufficiently from the clay; and a good many rich ore-deposits are not workable because of this fact.

If the ore-bearing material as delivered to the grizzly contains a considerable quantity of soft, nearly-decomposed sand-boulders, opportunity must be afforded to separate them from the material being delivered into the logs; for, when once in contact with the logs, they are quickly broken up and can then only be separated from the ore-product by jigging.

In connection with the delivery of ore-bearing material from the grizzly to the washer, the possibilities of a flume are often overlooked. A flume not only offers a cheap and convenient method for transporting the material between these two points, but is also of considerable advantage to the washer. In fact, some ores require no further treatment than to be made to travel several hundred feet down a gravity-flume. A metal-lined flume requires a minimum inclination of 11°.

IV. Washing.

The ordinary log-washer of the "ground-hog" variety is probably not a curiosity in any State of the Union; nor should we marvel at this, when we consider its wide usefulness, coupled with its history, which dates back to ancient Greece. The so-called "modern" washing-plant differs little from the earlier variety as regards log-design, the difference being mainly noticeable in the accessories that have been added; nor are the results obtained in these modern plants always entirely different from the results obtained in the less pretentious old ones. In too many instances, the modern plant is simply a copy of a neighboring successful one, and while the original design may have required all the extra trimmings, the latter plant did not; and it is quite possible that the trimmings effect a waste rather than a saving.

The flow-sheet for a complete plant would show the following:

All material is delivered into a revolving conical overhead screen. The oversize passes out at the end of the screen on to

a picking-belt, from which it is delivered into a crusher, thence on to another picking-belt, and then directly into the railroad-cars or storage-bins. Very often the crusher is omitted, and the material passes from the overhead screen on to a picking-belt, and is delivered to the loading-bins. The undersize from the overhead screen passes directly into the lower end of the log-washer. During the passage along the logs, most of the loose dirt is separated from the rock-material and flows out at the lower end of the logs with the water overflow. The rock-material passes out at the upper end of the logs and is delivered into a sand-screen. The oversize from the sand-screen passes on to a picking-belt for delivery into bins or railroad-cars. The undersize is either sluiced to jigs for treatment, or is carried away with the waste water.

As a rule, the proper functions of the various accessories do not receive proper study; and, when once installed, they are operated continuously, even though they may be responsible for useless waste. The work that each part of the washer-equipment may be expected to accomplish is not necessarily shrouded in mystery, and the limitations of each are easily determined.

Overhead Screen.—The principal object of this screen should be to eliminate from the washer-feed all material that is too large to be handled by the logs. It is true that in passing through this screen much dirt may be separated from the large boulders by means of numerous nozzles delivering water under pressure, but it is also true that this dirt would be more effectively removed could the dornicks be handled in the washer.

If the ore-material carries a large proportion of sand-boulders, the screen is of great advantage, since it cleanses these boulders and permits them to be easily detected by the picker-boys before they reach the crusher.

In some deposits the large lumps of ore are comparatively "close-grained" (in contrast to the honeycomb structure generally characteristic of ore-boulders), and a thorough rinsing is all that is required to make them marketable; with such deposits the overhead screen will increase the capacity of the washer considerably, as it can be arranged to handle all of the lump-ore, leaving only the fines for the logs.

If the ore-bearing material consists largely of loam and sand,

the overhead screen will be very effective. On the other hand, if the material consists of small ore-particles imbedded in clay, which tends to concentrate into clay balls, the screen cannot be depended upon to effect a separation, and may produce great waste of ore. The explanation is, that the clay, in "balling up," carries with it most of the ore-particles; and, since the balls produced are too large to pass through the screen, they are delivered from the end of the screen on the picking-belt and are then thrown away by the picker-boys. Or, if they are not thrown away, their clay-content will materially affect the resulting ore-analysis. The best way to overcome this difficulty is to abandon the overhead screen temporarily at least, and pass all of the material through the logs.

Log-Washers.—The function of the log is usually overestimated. Logs, if properly designed and erected, will eliminate from all classes of rock-material, clay, loam, and sand. They will not separate pyrites, quartz, and limestone from ore. Yet logs are continually being erected to handle material that consists of one-fourth ore and three-fourths useless rock.

Almost the only change effected in the design of logs during the past 1,000 years is the substitution, to a limited extent, of steel for wood. Each type has its advantages. The steel log is higher in first-cost, but in permanent plants its longer life will offset this difference. Its principal disadvantage is, that it does not permit the variations in lug-spacing so effectively employed with wooden logs when the character of the material handled changes suddenly. The principal disadvantage of the wooden log is, that it is liable to break if overloaded with large dornicks, and, at best, is short-lived. In some localities it is impossible to obtain timber suitable for logs.

In the manner of driving log-washers, recent practice has substituted intermediate friction-drivers to eliminate "breaking pins," or, worse still, breaking gears.

To obtain satisfactory results from any type of log-washer an adequate supply of water is an absolute necessity.

Crushers.—Crushers may be installed to accomplish any one of three things:

1. Reducing dornick ore to a size considered satisfactory by the furnace-manager who buys the ore. This may be anything from 1-in. ring to 6-in. ring.

2. Reducing dornicks of the honeycomb type to permit their being effectively handled in a log-washer. The cavities in ore of this type are filled with clay; and unless they are broken down into small sizes the clay cannot be eliminated by the washer.

The following figures show the effect of installing a crusher between grizzly and washer at a plant operated by me several years ago.

Average of Ore Loaded in Railroad-Cars.

Before Installing Crusher.

	Metallic Iron. Per Cent.	Silica. Per Cent.	Alumina Per Cent
Aug. 4,	44.33	16.32	4.02
Aug. 11,	48.37	11.44	3.09
Aug. 19,	44.48	16.80	4.28
Aug. 26,	46.74	13.90	4.30

After Installing Crusher.

	Metallic Iron. Per Cent	Silica. Per Cent.	Alumina. Per Cent
Sept. 4,	48.21	12.04	3.61
Sept. 11,	49.11	10.76	3.60
Sept. 18,	47.77	12.50	3 80
Sept. 25,	49.80	11.52	3.78

3. Crushing breccia dornicks, consisting of loosely-cemented fragments of rock and ore, to a size that will allow them to be fed into jigs. Deposits that would be materially benefited by such treatment are rare, and very careful experiments should be made before installing a crusher for such duty.

Sand-Screens.—Most of the sand contained in the material delivered to the log-washer passes along the logs and is delivered with the ore-product. This has led to the practice of passing all material from the washer-discharge into a revolving screen made of wire cloth about $\frac{1}{8}$-in. mesh. A stream of water playing on the inside of this screen forces most of the sand out. It also forces all the ore fines out; and, in many plants, investigation reveals the fact that the material passing through the sand-screen is superior to that recovered. Constant sampling of the tailings from any washer is always to be recommended. If the tailings contain a large percentage of ore, and yet are too siliceous to be marketable, the feasibility of recovering the ore by means of concentrators should be investigated. When jigging is employed, all of the material

passing through the sand-screen should be delivered to the jigs for treatment. This permits variations in the size of openings in the sand-screen; and careful experiments are required to determine just what size of material should be allowed to pass through the screen into the jigs. In most deposits, material that will be rejected by screens having 1.5-in. perforations is not materially benefited by jigging.

Picking-Belts.—Picking-belts, as the name implies, are slow-moving conveyors of any description that afford opportunity to pick out clay balls and worthless rock from the ore-product as delivered by the washer. If the ore is crushed before being delivered to the washer, picking-belts are generally installed to feed the crusher. To be effective, all of the material on the belt must be thoroughly rinsed by numerous sprays before passing the picker-boys; otherwise the material is liable to be so covered with mud that it is impossible to distinguish between the ore and refuse. Picking-belts are worthless unless manned by a sufficient number of competent pickers to separate the waste material during its passage between washer and bin; yet such belts are often turned over to one or two boys without further thought or supervision.

The greatest opportunity afforded for improvement in the methods now practiced at brown-ore washers would seem to be in connection with these picking-belts. Attachments making it possible to replace with mechanical separation the present unreliable hand-picking will undoubtedly be perfected in the near future.

V. Concentration.

For concentrating brown ore (beyond the results that may be obtained in a washer) three means are available, although only one (jigs) has been adopted to any great extent. I refer to jigs, reciprocating tables, and magnetic and electrostatic separators.

The effectiveness of concentrators is dependent on three things: the size of the ore; the material with which the ore is associated; and the quality of the ore.

If the crude ore, as delivered on board cars, contains a large amount of fines, say from 1 in. down, and it appears that these fines contain a considerable amount of siliceous material, the substitution of a 1-in. mesh screen in place of the fine mesh

standard sand-screen, and the subsequent concentration of the fines from the screen, will probably be advisable.

If the ore-product consists largely of breccia, made up of sand, rock, and ore, crushing and subsequent concentrating may eliminate the siliceous contents.

If the ore-product contains a large amount of "galvanized" sand-rock, which cannot be eliminated by picking-belts on account of its resemblance to the ore-mass, crushing and subsequent concentrating may be required.

The waste from the ordinary sand-screen may carry enough ore to justify the erection of a concentrator, even though none of the rest of the product requires it.

Before installing a concentrator in connection with any plant, complete tests should be required. Such tests in connection with jigs may reveal the fact that the specific gravity of the ore and rock is so similar that jigs are not effective; again, tests in connection with electrostatic separators may prove them worthless for the purpose intended, because of the large silica-content of the ore to be treated.

Magnetic and electrostatic separation has received very little attention from brown-ore operators up to the present time.

The advantages of the reciprocating table have been also quite generally overlooked.

The Sintering of Fine Iron-Bearing Materials.

BY JAMES GAYLEY, NEW YORK, N. Y.

(Wilkes-Barre Meeting, June, 1911.)

THE paper presented to the Institute in 1910, by H. O. Hofman, on Recent Progress in Blast Roasting,[1] has called the attention of the iron industry to the adaptability of these processes to the reclamation of by-products such as flue-dust and blue billy, and the better preparation of concentrates and fine ores for use in blast-furnaces.

The waste of valuable iron-ore through the production of flue-dust has increased at an enormous rate, and much of it has in

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[1] *Trans.*, xli., 739 (1911).

reality been wasted as far as future recovery is concerned. Only in recent years has a possible future value been recognized, and the material been stored. The increased use of Mesabi ores, which carry considerable fine ore, is principally responsible for the great increase in the production of flue-dust. The amount of fine material that is carried over depends on the fineness of the material and the velocity of the gases, and also to a very great extent on the regularity or irregularity of the working of the furnace. Attempts to recover a part of this loss have been made by recharging a portion of the production into the furnace, but as this material has been once carried out from the furnace, it is naturally in good shape to be carried out again.

A recent practice at some works, is to soak the flue-dust thoroughly with water to give it more cohesiveness, but by many this is considered of doubtful advantage in the furnace, and the gases are in consequence heavily laden with moisture.

There are vast deposits of magnetic iron-ores in the United States and Canada that are too low in iron for use at the present time, but which can be economically concentrated into very rich material; in many cases the fineness of crushing necessary to secure proper concentration has prevented their use except in extremely limited quantities. The reclamation of these ore-bodies will add tremendously to the ore-reserves of the United States, and this can best be done by a simple and efficient method of sintering.

Attention was directed especially to the Dwight and Lloyd system of sintering fine material in thin layers by internal combustion as promising to solve this problem most efficiently. The Dwight and Lloyd patents cover most of the simple forms of apparatus by which their process can conveniently be carried out, but the one that has given the best satisfaction in practice and has now been adopted as the standard is known as "Type E," or the "straight-line-conveyor type" described by Hofman. As shown by Fig. 1, the machine consists essentially of a frame of structural steel supporting a sheet-iron suction-box, open at the top, over which may be pushed a train of conveyor-elements called "pallets," each of which has a floor composed of ordinary herring-bone grates, and which slides on its planed bottom, making an air-tight joint with the horizontal top edges

of the suction-box on which it rests. The vertical surfaces of contact of the pallets with each other are also accurately planed, so that all joints are closed air-tight when the train of pallets is being pushed along.

An exhaust-fan, connected with the suction-box by suitable piping, induces air-currents to travel downward through the openings in the pallet-grates and through the permeable material resting upon them. To trap the air properly, a smooth-surfaced dead-plate, somewhat longer than one pallet-length, is bolted to each end of the suction-box.

The movement of the train of pallets is accomplished by a pair of cast-steel sprocket-wheels, which serve the double purpose of lifting the pallets from the lower level and pushing them horizontally across the suction-box. Each pallet is provided with four small roller-wheels which hang idle while the pallet is traveling over the suction-box, but serve to carry the pallet on its return trip to the point of beginning. The return of the pallets is provided for by a pair of semi-circular discharge-guides, terminating in a lower track-way sloping downward to the base of the main sprocket-wheels, and continuing as semi-circular guides around their peripheries. The pallets, at the completion of their journey across the suction-box to the point of discharge, have their wheels engaged by the curved guides, and when pushed still further, beyond the crest of the curve, break away from the train that is pushing them, and one by one, drop with a sharp blow on the upturned edge of the pallet just preceding. This shock serves to dislodge the cake of sinter from the surface of the grates, which now stand more or less vertical. The train of discharged pallets, in the guides and on the inclined lower track-way, crowds down by its own weight to the foot of the main sprocket-wheel. During this period of their travel the pallets are upside down, which automatically tends to clean out the grate-slots. The sprocket-wheels lift the train of pallets to the upper level and the cycle is completed.

We thus have a practically endless conveyor, any individual element of which can be removed for repairs and a new one substituted without stopping. The circuit may, if desired, be made a closed one, and this arrangement has been used under special conditions; but, in general, it is customary to leave an

interval in the train of about one and a half pallet-lengths, which gives just about the right amount of shock.

The speed of horizontal travel of the pallets is adjustable to

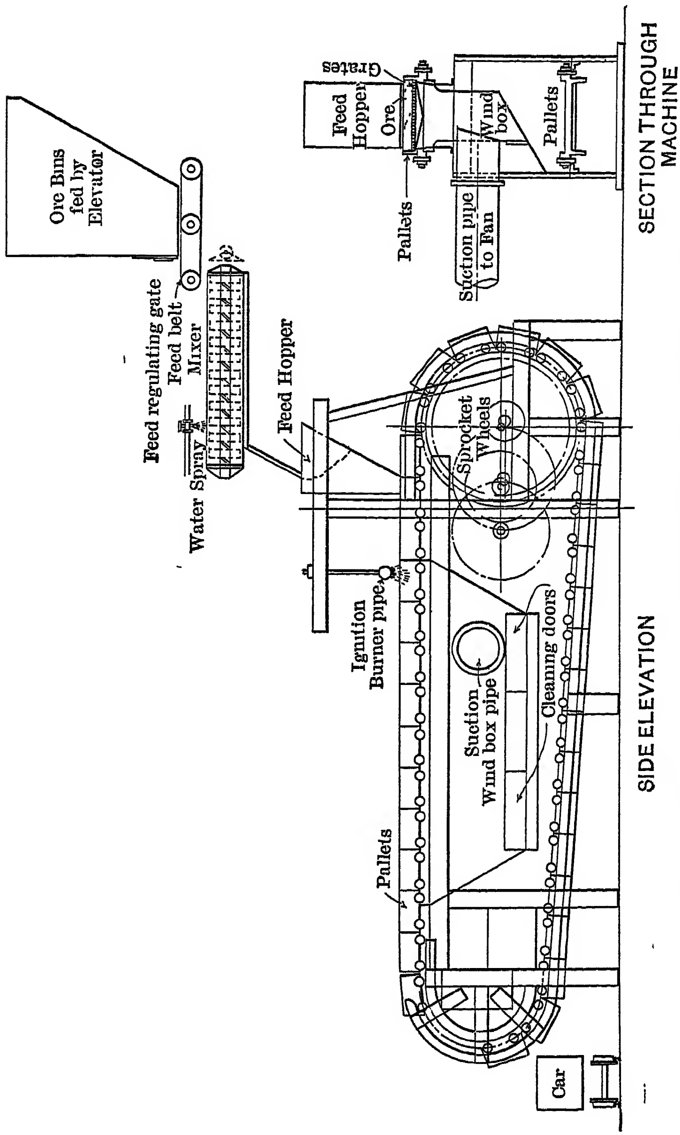

Fig. 1.—Dwight and Lloyd Sintering-Machine, Conveyor Type.

meet varying requirements, with the usual range from 7 to 30 in. of linear travel per minute.

The ore-charge is automatically fed to the pallets in a thin

layer (from 4 to 6 in. thick) from a simple funnel-shaped hopper of the same width as the pallets, hung directly over them at a point between the main sprocket-wheels and the suction-box. There being no bottom to the hopper, the material rests directly on the pallets and is dragged out by their movement, the front edge of the hopper acting as a scraper to form a uniform layer of the proper thickness.

The stream of ore emerging from the hopper passes under an igniting-device which kindles the combustible elements in the charge on its top surface, and the combustion thus started is carried downward through the mass by the air-currents while the material is passing over the suction-box. This ignition can be accomplished by almost any kind of flame that will give a quick, intense heat. The amount of heat required at this point of the operation is exceedingly small and the cost of ignition correspondingly low. The wide variety of suitable means makes it possible to meet almost any local requirements.

The complete cycle of operations is as follows: A pallet being pushed outward tangentially from the top of the sprocket-wheels, passes under the feed-hopper, where it takes its load of ore in the form of a continuous, even layer of charge, say 4 in. thick. It next passes under the ignition, where the top surface is ignited, and at the same time the charge comes within the influence of the down-draft induced through the suction-box by the exhaust-fan. The air-currents promote rapid internal combustion of the fuel ingredients in the charge, and carry the action progressively downward from the top surface until it reaches the grates. This internally-developed heat and the chemical reactions consequent thereto, serve to bind the mass together until it becomes a coherent cake of cellular material, much resembling coke. The speed of the machine should be regulated so that the combustion- and sintering-operation is complete when a given pallet has reached the far end of the suction-box, where the cake is discharged by the pallet dropping into the discharge-guides and striking the one just preceding it. The empty pallets then gradually crowd back to the face of the sprocket-wheels, are slowly raised to the upper track, take their load, and make a new trip.

This type of machine is now made in two standard sizes, one having a suction-box 30 in. wide by 150 in. long and a nominal

rated capacity of 50 tons per day; and the other with two suction-boxes in tandem, having a width of 42 in. and an aggregate length of 264 in., and having a nominal rated tonnage of 100 tons per day on average material. The area of the suction-box is the measure of the capacity of the machine, and the suction-fan must be so proportioned as to maintain a vacuum of about 6 oz. when handling approximately 4,000 cu. ft. of gases per minute, this being the average volume from each 100-ton unit.

Such a fan, with short and straight pipes and running at about 850 rev. per min., requires from 25 to 35 h-p. The sintering-machine itself consumes about 1.5 h-p., but 10 h-p. is usually allowed for machine, conveyors, feeds, and mixers—in fact, everything except the fan.

Each sintering-unit is self-contained and occupies space approximately as follows: 30- by 150-in. machine (so-called 50-ton unit):

Length over all, 27 ft.
Width, . . 7 ft.
Height of top of hopper above foundation, 11 ft. 4 in.
Units in battery may be set with 11-ft. centers.
Weight of complete machine, approximately 16 tons

42- by 264-in. machine (so-called 100-ton unit):

Length over all, 40 ft. 8 in.
Width, . . 7 ft. 6 in.
Height of top of hopper above foundation, 13 ft. 9.5 in.
Units in battery may be set with centers 12 to 14 ft. apart.
Weight of complete machine, approximately 26 tons.

The grates are of the simple herring-bone pattern and are made of cast-iron. There should be very little breakage. The heat developed in the operation being internal to the ore-mass, does not cause the pallets to become very hot, and there is but little damage from this source. Moreover, on account of the extremely slow movement of the mechanism, the wear and tear is very small. In one plant which was in steady operation for two years the average cost of supplies and repairs was from 2 to 4 cents per ton; 5 cents per ton will easily cover all ordinary contingencies. In many ways the excellence of this particular type of mechanism has been thoroughly demonstrated, and it may now be confidently stated that it is a simple, effi-

cient, and workmanlike device for carrying out this special purpose, and can be adapted to almost any location.

A number of iron-bearing materials, of different kinds, were treated on this machine, and in each case with satisfactory results. Among them were two shipments of iron flue-dust, which were widely different as to physical condition. One was the usual character of flue-dust, which I shall designate as No. 1, while the other, No. 2, was extremely fine, 50 per cent. of it passing through a 100-mesh sieve; but the sintered product of each was not distinguishable, and both were ideal in size and structure for the blast-furnace. There were no large and compact masses like the product from the briquetting-process, nor was the material rolled together in balls from the size of a pea to that of a cannon-ball, as in the revolving-kiln; but, instead, the individual pieces were cellular, like open pumice-stone or porous cinder, which helps materially towards economic reduction in the furnace, as a large area of contact is provided between the ore and gases.

In Schinz's book,[2] published in 1871, a chapter is devoted to "area of contact." The opening sentence is as follows:

> "A chemical action can only take place between two bodies, however great their affinity, if they are in intimate contact with each other; and the rapidity of this action will be so much greater, the more numerous the points of contact are."

In the Dwight and Lloyd method of sintering with a bed of material that is not disturbed or agitated during the sintering-operation, the sintered product is all so cellular that a large "area of contact" is provided; and its reducibility is very great compared with the more massive agglomerated products, just as coke, by reason of its cellular spaces, burns more readily than anthracite coal, which can have only a superficial combustion.

Although the product from the Dwight and Lloyd furnace in sintering flue-dust is of a desirable size for blast-furnace use, yet a fair proportion of the product would be suitable for use in the open-hearth furnace.

In sintering materials which do not contain any heat-producing substances, recourse can be had to the practice of the

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[2] *The Action of the Blast-Furnace.*

ancient Catalan or Corsican process, where carbon fuel was mixed with the ore, and which, in its first stage, was an agglutinizing process. In order to test the machine on this class of work, some magnetic concentrates were treated, after being mixed with 7 per cent. of coal, and the product was found to be satisfactory in every particular. The material was sintered into a coherent mass, but so open and cellular in structure that the mass, in discharging from the pallets, broke into very convenient sizes for the furnace, and without any fines. As the mixture contained less carbon than the flue-dust, it was sintered much more quickly. While in the test on flue-dust, a travel of 12 ft. in the grate-movement was required to complete the sintering, the concentrates were completed in a travel of 6 ft. This represents, in the treatment of magnetic concentrates, a greatly-increased capacity for the machine.

Some Cuban (Mayari) iron-ore was also treated on the machine after being mixed with 7.5 per cent. of coal and coke in alternate tests, and afterwards the ore was mixed with 10 per cent. of coal in one test and 10 per cent. of coke in another; but the use of 10 per cent. of fuel did not show any advantage over 7.5 per cent., nor were the results from coke any better than from coal. The sintered product resembled closely that obtained from the flue-dust; there was very little fine material, and, in fact, no fines that would require re-treatment. The sintered material was irregular in shape, with an average size of a hickory-nut.

The following are analyses of the material treated:

Sample.	Fe. Per Cent.	P. Per Cent.	Mn. Per Cent.	SiO_2. Per Cent.	Al_2O_3. Per Cent.	CaO. Per Cent.	MgO. Per Cent.	Carbon. Per Cent.
No. 1. Flue-dust, . .	46.06	0.194	0.54	9.68	3.00	1.80	0.80	17.00
Sintered product, .	57.90	0.260	0.66	12.30	3.95	2.00	1.20	0.60
No. 2. Flue-dust, . .	46.43	0.123	0.60	9.88	2.72	2.00	1.44	13.75
Sintered product, .	58.84	0.150	0.75	11.81	3.05	2.50	1.71	2.10
Magnetic concentrates,	57.52	0.090	0.56	9.70	3.43	0.35	0.10	0.00
Sintered product .	59.65	0.110	0.60	10.60	4.00	0.30	0.10	0.00

Sulphur.

	Per Cent.
Magnetic concentrates with 7 per cent. of coal, . . .	1.17
Sintered concentrates, ...	0.006

Sieve-Test.

Sieve.	No. 1. Flue-Dust Per Cent	No 2 Flue Dust. Per Cent.	Magnetic Concentrates. Per Cent
On 10-mesh,	14.0	4.0	28 0
On 20-mesh,	31.0	1.0	44.0
On 40-mesh,	31.0	6.0	15.0
On 60-mesh,	14.0	4.0	7.0
On 80-mesh,	3.0	15.0	2.0
On 100-mesh,	3.0	20.0	1.0
Through 100-mesh,	4.0	50.0	3.0

	Ferrous Iron Per Cent	Ferric Iron Per Cent.	Total Iron. Per Cent
Cuban (Mayari) ore (dried at 212°),	0.63	47.80	48.43
Sintered product,	9.67	44.30	53.97

Sieve-Test, Mayari Ore, Sintered.

	Per Cent
On 2-mesh,	53.88
On 4-mesh,	16.33
On 8-mesh,	23.35
On 20-mesh,	4.33
On 40-mesh,	1.12
On 60-mesh,	0.51
On 80-mesh,	0 02
On 100-mesh,	0.14
Through 100-mesh,	0.32

The physical structure of the sintered product varies under different conditions. Where there is a large amount of moisture and carbonaceous matter present, a corresponding shrinkage within the mass must take place as the volatile constituents are driven out, and this may cause the cake of sinter as a whole to break up into irregular-shaped masses or fingers. The smallest of these pieces, however, have a cellular structure like pop-corn, which is peculiarly desirable for the blast-furnace. In the case of magnetite concentrates, where there is less internal shrinkage, the sinter comes off in slabs having an open structure.

The Cuban ore being the finest, the sinter was of a smaller average size than the magnetic concentrates, which were coarser and did not shrink so much in sintering. The flue-dust being coarser than the Cuban ore produced a sinter about midway in size between the flue-dust and the concentrates. The cohesiveness of the material is inversely as the amount of internal shrinking of the mass during sintering.

Among the advantages observed in the Dwight and Lloyd process, the following may be noted:

1. The feeding of material to and discharge from the machine, without interfering with the continuity of the process.

2. The down-draft of air exerts pressure in the direction of the gravity of the mass, and prevents the displacement of particles.

3. The down-draft of air intensifies the combustion at the beginning of the sintering, and towards the end of it operates efficiently to cool the mass.

4. The sintering-operation is under constant observation during the whole period, and permits of immediate changes in adjustment.

5. The process can be stopped at any time to make any changes without interfering with or clogging any part of the apparatus.

6. The duration and activity of treatment are subject to ready control.

7. The adjustability of the process makes it adaptable to treating any class of fine material, without modifying the construction.

8. The withdrawing of the gases by a fan provides a heating medium for drying ores carrying a surplus of moisture.

9. There is no nodulizing of the sintered material, and the cellular structure, which is so important, is preserved.

10. The product is ideal in structure for use in the blast-furnace, on account of the large "aiea of contact" provided, and its adaptability in size for distribution in and passage through the furnace.

With the large productive capacity that has been built up in the iron and steel industry in the United States, matters of economy in production are now engaging the attention of the industry to a much greater extent than in the past. The most promising field for effecting economies therein, is in the manufacture of the basic metal—pig-iron; in which several avenues are still open for effecting great reductions in cost. A very important field to operate in, is the treatment of the fine ores before being charged into the furnace. Twenty-five years ago the practice of charging large lumps of ore and stone and large pieces of fuel was discontinued, and the crusher came into gen-

eral use for reducing these materials to a more uniform size, and with beneficial results in the furnace. The increasing use of the Mesabi ores has led to the other extreme in practice, so that the fine ores, and the fine dust resulting from their use, require an agglomerating process, in order to return to the ideal condition of material as it was "sized" by the crusher.

The use of very fine materials in the blast-furnace has not been successfully worked out, and probably never will be in the modern blast-furnace, and no time should be lost in adopting efficient and economical methods for treating these materials to make their use successful. The practice of recharging the flue-dust as such, is considered by many a questionable one. Some furnace-men hold the opinion that while the recharged flue-dust is retained in large part in the blast-furnace, it is nevertheless detrimental, as it tends to collect on the bosh-walls, and causes frequent slips and irregular working. Because so much ore is saved from the waste, it does not follow that it represents a saving in cost of pig-iron. The screening of coke to eliminate the fine pieces is certainly beneficial, but it does not seem logical to recharge the same kind of material when intermixed with fine ore, as in flue-dust, into the furnace.

The actual amount of objectionable "fines" in the Mesabi ores represents only a small percentage; but its pernicious influence is out of all proportion to the amount involved. At some furnaces in England, where the fine material is screened from the ores and sintered, very beneficial results have been obtained.

When the screening of the fine material from the coke was first advocated, it was objected to by many, as representing a waste of fuel, although of poorer quality, that might have some value in the furnace; but now the economic value of the practice is fully appreciated. The same practice applied to ores, and sintering the fine material to prevent waste, promises as great or even greater benefits.

The Fuel-Efficiency of the Iron Blast-Furnace.

BY JOHN JERMAIN PORTER, CINCINNATI, O.

(Wilkes-Barre Meeting, June, 1911.)

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I. INTRODUCTION.

The heat-balance of the blast-furnace has been a favorite subject for discussion for many years, and its study has contributed much to our knowledge, and still possesses a certain academic interest. As a means of accounting for differences in fuel-consumption, however, it has failed utterly, and it is evident that we must look elsewhere for the means of accomplishing this end.

In my opinion, the explanation of the fuel-requirements involving the conception of heat available and necessary above a critical temperature, as advanced by Johnson[1] and elaborated by Howe, Raymond and others,[2] affords a theoretically correct basis for such an accounting. It is the purpose of this paper to set forth a first crude attempt to construct a formula which shall show quantitatively the relation between the various factors affecting fuel-economy, and afford a means of comparing the enormous amount of data on the fuel-consumption of various blast-furnaces, at various times and under various conditions of operation. I have also applied this formula to a large number of individual cases, and have, by its means, compared the furnace-practice of several important iron-making districts.

II. Derivation of Formula for Fuel-Requirements.

The general expression for the fuel-requirements of the blast-furnace, which I believe to be theoretically correct as to form, is as follows:

$$\text{Carbon per ton of iron} = \frac{\text{Heat per ton of iron necessary in hearth above } Tc.}{\text{Heat per pound of carbon available in hearth above } Tc.}$$
$$\times \text{ Factor of rate of driving.}$$
$$+ \text{ Carbon dissolved by } CO_2 \text{ of flux.}$$
$$+ \text{ Carbon dissolved incidental to reduction of ore.}$$
$$+ \text{ Carbon dissolved in pig-iron.}$$

(Note.—Tc = critical temperature.)

Since, according to Johnson's theory, fuel-economy is usually limited by heat available and necessary in the hearth, rather than by the total heat supplied and necessary, we may write

$$\text{Carbon needed in hearth} = \frac{Hn}{Ha}$$

in which Hn is the heat per ton of iron necessary in the hearth above some critical temperature, and Ha is the heat per pound of carbon available in the hearth above this same critical temperature.

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[1] *Trans.*, xxxvi., 454 (1906).

[2] *Trans.*, xxxvi., 792 to 798 (1906); *Trans.*, xxxvii., 216 to 237 (1907).

1. *Heat Available in the Hearth.*

a. Method of Calculation.—The heat available is equal to:
Heat of combustion of carbon to CO = 4,380 B.t.u.

+ Heat brought in by blast = weight of blast per pound of C × specific heat × temperature.
+ Heat brought into hearth by C = 1 × specific heat of C × *Tc*.
— Heat to dissociate moisture of blast = 5,800 × pounds of water per pound of C.
— Heat carried out of hearth in gases of combustion = weight of gases per pound of C × specific heat × *Tc*.

NOTE.—The origin of this expression may perhaps be more clearly understood if we regard the fusion-zone of the blast-furnace as a definite space, Fig. 1, receiving heat from four sources, and, besides what is utilized, losing heat from three sources, as follows:

Heat Supplied	Heat Lost.
By combustion of carbon.	Carried out in gases.
In hot blast	Carried out in iron and slag.
Sensible in carbon	Absorbed by decomposition of moisture.
Sensible in iron and slag.	

All materials leaving this zone are assumed to pass out at the critical temperature, and all solid materials entering the zone must, theoretically at least, enter at the critical temperature because of counter-current transfer of heat with the gases. Hence, the heat received in the slag, plus iron, is balanced by heat carried out in the slag, plus iron, and *Ha* is equal to the heat of combustion of carbon, plus the sensible heat brought in by the carbon, plus the heat brought in by the blast, minus the heat necessary to decompose the moisture of the blast, minus the heat carried out by the gases of combustion.

This formula is only the expression of the method of calculation previously used by Johnson,[3] but in applying it I have used different, and, I believe, more accurate values for the several constants. The values used, all in B.t.u. and Fahrenheit degrees, are as follows:

Specific heat of N and CO $= 0.2405 + 0.0000117t$.
Specific heat of O $= 0.2104 + 0.0000102t$.
Specific heat of H $= 3.70 + 0.0001667t$.
Specific heat of H_2O vapor $= 0.42 + 0.000103t$.
Specific heat of C $= t \times \left(0.5 - \frac{220}{t}\right)$ (t over 1,800°).

Heat of combustion of C to CO = 4,380 B.t.u. per lb
Heat of decomposition of water-vapor = 5,800 B t.u per lb.

[3] *Trans.*, xxxvi., 476 (1906).

These values are for the most part taken from Richards's *Metallurgical Calculations*,[4] and represent the most recent and authoritative opinion on the subject.

Since the calculation of *Ha* involves much labor on account of the variation in specific heats and in weights of blast and gases, I have worked out each value for a sufficient number of cases, using all these refinements, and have, from these data,

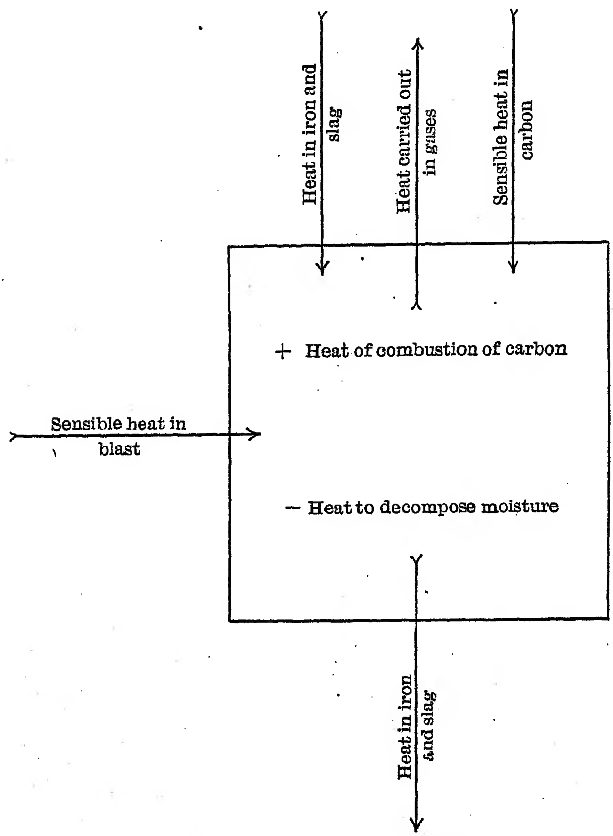

FIG. 1.—SOURCES OF HEAT-SUPPLY AND HEAT-LOSS IN THE FUSION-ZONE OF A BLAST-FURNACE.

plotted Fig. 2, which affords a ready means of obtaining *Ha* for any given set of conditions. To get *Ha* by the use of this diagram, it is only necessary to know three things: temperature of blast, moisture in blast in terms of grains per cubic foot (of moist air at the dew point), and the critical temperature.

[4] Part I., Chapters 2, 4, and 5 (1906).

b. Data for the Determination of Critical Temperature.—The first of these factors is, of course, invariably a matter of record, and while the same is not true of the moisture of the blast, still the interest aroused by Gayley's invention is causing a

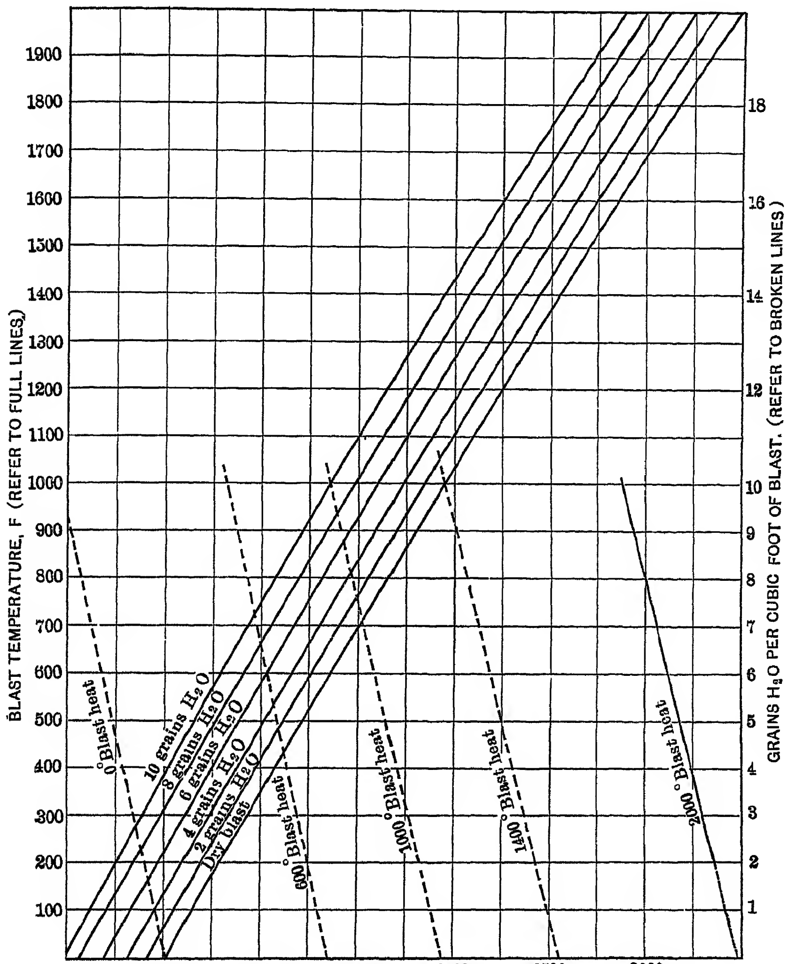

FIG. 2.—HEAT AVAILABLE PER POUND OF CARBON FOR VARYING TEMPERATURE AND HUMIDITY OF BLAST. CRITICAL TEMPERATURE, 2,700° F.

rapid accumulation of data on this point. The critical temperature, however, is a stumbling-block, for not only are our data very scant, but it is even difficult to define the term accu-

rately. Thus far, the most satisfactory conception is the original one given by Johnson,[5] who says:

"It thus becomes evident that the temperature necessary, not only to melt the cinder, but to make it sufficiently fluid to perform its functions properly, is the 'critical temperature,' since the slag and iron must be brought to this temperature, and the final reduction of the ore must be performed above it (and, most probably, other reactions). What may for convenience be called the free-running temperature of the cinder is therefore taken in this paper as the critical temperature.

The following data embrace all the information bearing on this subject that I have been able to find in the limited time available.

Johnson,[6] by observations with a Mesuré and Nouel pyrometric telescope, determined the critical temperature of a Virginia coke-furnace making basic iron to be 2,750°.

A series of observations taken by myself with a Mesuré and Nouel pyrometric telescope on the outflowing slags of a group of Northern furnaces gave the following results:

4 furnaces making Bessemer iron,	range of slag-temperature, 2,300° to 2,660°.
1 furnace making basic iron,	range of slag-temperature, 2,282° to 2,507°.
1 furnace making spiegel,	range of slag-temperature, 2,572° to 2,732°.

A series of observations with a Wanner optical pyrometer on the temperature of the outflowing slag of two Alabama furnaces, both making foundry-iron, resulted as follows:[7]

Range on the one furnace, ...	2,100° to 2,300°.
Range on the other furnace, ...	2,460° to 2,470°.

Linville[8] determined the temperature of the outflowing slag of a Pennsylvania furnace by means of a Fèry optical pyrometer. The furnace was making foundry-iron. The temperature range was from 2,570° to 2,680°.

Le Chatelier, quoted by Turner,[9] gives the temperature of exit of gray Bessemer iron at the end of a cast as 2,858°.

Several of these determinations are open to more or less doubt, since, as is well known, the various forms of optical pyrometers may give quite erroneous results, unless used under

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[5] *Trans.*, xxxvi., 481 (1906).

[6] *Trans.*, xxxvi., 472 (1906).

[7] Oral communication from Mr. Banks Hudson.

[8] *Bulletin* No. 39, March, 1910, p. 245.

[9] *Metallurgy of Iron*, 2d ed., p. 144.

carefully standardized conditions. It seems probable, however, that the critical temperature of the average coke-furnace varies between 2,600° and 2,700°, and, rightly or wrongly, I have chosen the value of 2,700° for use in the greater number of my calculations.

2. *Heat Necessary in the Hearth.*

The heat necessary above the critical temperature embraces the items of:

The fusion and superheating of the slag.

The fusion and superheating of the pig-iron.

The reduction of the silicon.

The reduction of the last traces of ferrous oxide.

Some other items, including loss of heat in cooling-water, losses by radiation, reduction of the other metalloids, etc.

It is obviously impossible in the present state of our knowledge of blast-furnace phenomena to place numerical values on all of these items, and opinions will vary as to how much detail is desirable here.

a. Heat to Care for Slag.—The heat necessary to care for the slag in the hearth of the furnace is theoretically approximately equal to the latent heat of fusion of the slag plus the heat contained in it above its melting-point. From data given by Richards,[10] we find that the specific heat of an average slag is 0.2874 $(1 + 0.00039t° \text{ C.})$. Assuming the melting-point as 1,300° C., the heat in the slag at the melting-point is 374 cal. The total heat in the slag coming from the blast-furnace has been determined by many experimenters, the values given ranging from 400 to 570 cal. Accepting the value used by Bell of 550 cal., the heat in the slag above the melting-point will be equal to $550 - 374 = 176$ cal., or 317 B.t.u., per pound.

So much for theory. We may check this roughly by reference to practice.

Hartman[11] says that for each pound of slag 0.31 lb. of coke containing 88 per cent. of fixed carbon is needed when using from 1,200° to 1,300° of blast-heat. These figures correspond to *Ha* equal to about 1,800 B.t.u., and 0.2728 lb. of carbon, or 491 B.t.u. per lb. of slag.

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[10] *Metallurgical Calculations,* Part I., p 115 (1906).

[11] *Journal of the Franklin Institute,* vol. cxxi., No. 5, p. 332 (May, 1886).

Forsythe[12] says that 1 lb. of slag requires about 0.25 lb. of carbon to melt it. This probably refers to Northern practice where *Ha* averages about 1,400 B.t.u., and if so, is equivalent to 350 B.t.u. per pound of slag.

The agreement between theory and practice is very poor, although the value of 400 B.t.u. per pound of slag is indicated as being somewhere near the truth.

b. Heat to Care for the Pig-Iron.—The heat required to melt and superheat the pig-iron may be calculated in a similar manner, and is found to be in the neighborhood of 400,000 B.t.u. per ton of pig-iron.

c. Heat to Reduce Silicon.—The heat required to reduce the silicon may be calculated on the basis of theory. Richards[13] gives $Si + O_2 = 180,000$ cal., which is equal to 259,201 B.t.u. per 22.4 lb. of silicon (1 per cent. of Si in 1 ton of pig). Elsewhere[14] he regards the value $Si + O_2 = 196,000$ cal., equivalent to 282,240 B.t.u. per 22.4 lb. of silicon, as more probable.

From practice we get the following results:

Forsythe[15] says that each pound of silicon above 1 per cent. requires an additional 5 lb. of carbon. This probably refers to Northern practice with *Ha* equal to approximately 1,400 B.t.u. Hence, we have $5 \times 22.4 \times 1,400 = 156,800$ B.t.u. per 22.4 lb., or 1 per cent., of silicon.

Meissner[16] says that in practice 0.12 per cent. variation in silicon is found to be brought about by 1 per cent. change in burden. This is the same as saying that 1 per cent. increase in fuel per ton of iron equals 0.12 per cent. increase in silicon. With Northern practice 1 per cent. of the fuel per ton of iron is approximately 22 lb., the fixed carbon in the fuel is about 88 per cent., and *Ha* is about 1,400 B.t.u. Hence, we have $22 \times 0.88 \times 1,400 \div 0.12 = 225,867$ B.t.u. per 22.4 lb., or 1 per cent., of silicon.

Johnson[17] says that it requires at least 20 per cent. more fuel to make iron with 3.5 per cent. of silicon than for iron

[12] *The Blast Furnace and the Manufacture of Pig Iron*, p. 48 (1908).

[13] *Metallurgical Calculations*, Part I., p. 15 (1906).

[14] *Idem*, Part II., p. 267 (1907).

[15] *The Blast Furnace and the Manufacture of Pig Iron*, p. 48 (1908).

[16] *Trans.*, xxxvii., 212 (1907).

[17] *Trans.*, xxxvi., 482 (1906).

with 1.5 per cent. of silicon. Applying this statement to an average practice represented by 2,500 lb. of coke per ton of iron, 88 per cent. of fixed carbon, and *Ha* equaling 1,500 B.t.u., we have $2,500 \times 0.20 \times 0.88 \times 1,500 \div 2 = 330,000$ B.t.u. per 22.4 lb., or 1 per cent., of silicon.

From these widely-divergent figures I have chosen the value of 300,000 B.t.u. per 22.4 lb., or 1 per cent., of silicon as giving good results.

d. Heat for Other Items.—On the other items going to make up the heat necessary in the hearth there are but few data. Bell [18] gives for one furnace the loss of heat in the cooling-water as 343,600 cal. per hour, or about 600,000 B.t.u. per ton of pig, but it is needless to state that these figures have very little significance as regards present conditions of furnace-practice. Langdon in some calculations of the heat-balance of blast-furnaces [19] finds by difference that the total losses by radiation and cooling-water for a number of furnaces varied between 1,200,000 and 3,100,000 B.t.u. per ton of pig. These losses, of course, vary with the furnace construction and output, and, although they are probably far from constant in practice, I can see no way to do other than to lump them in a constant in this present investigation.

There are a few other items entering into *Hn*, such as the heat absorbed in the expansion of the blast, the heat of reduction of phosphorus, manganese, and of the lime present as CaS, but these items, being relatively unimportant and nearly constant, may be included in the constant. The heat required to reduce the last traces of iron oxide will be discussed a little later under another head.

3. *The Effects of Rate of Driving.*

It is a well-known fact that fuel-economy is largely affected by the rate of driving, and former volumes of our *Transactions* [20] contain much discussion of this phase of the subject; notwithstanding which, no quantitative laws have yet been deduced and

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[18] *The Chemical Phenomena of Iron Smelting* (1872).

[19] *Trans.*, xl., 614 (1910).

[20] See more especially James Gayley, The Development of American Blast-Furnaces with Special Reference to Large Yields, *Trans.*, xix., 932 (1890–91), and the discussion of this paper in this and following volumes.

the correct rate is still a matter of doubt, except as it has been determined by experiment in individual cases.

According to theory, increase in rate of driving should increase fuel-consumption in two ways:

1. Through rendering less perfect the heat-transfer from the hot gases rising from the hearth to the relatively cool solid materials descending to the hearth.

2. Through imperfect reduction of the ore, which, if it enters the hearth, undergoes the reaction $FeO + C = Fe + CO = - 36{,}540$ cal., or there is an absorption of 14,616 B.t.u. for each per cent. (22.4 lb.) of iron reduced in this place in addition to the loss of carbon. This heat-absorption is a direct charge upon *Ha*, and hence is a far more serious matter than when the same reaction takes place at a higher level.

Both of these factors depend upon other things besides the actual pounds of air blown into the furnace. The heat-transfer depends upon the relative weights of the descending solids and the ascending gases; in other words, on the burden-ratio, as well as on the velocity of the gases. The perfection of reduction depends in part on the density of the ore and fuel, on the height of the furnace, and on the reducibility of the ore, since these factors, as well as the rate of driving, determine the time of exposure actual and necessary.[21]

In selecting values for this factor of rate of driving, I have been at a loss for means of expressing all of the factors, and have finally decided that the best that can be done at present is to take this factor as proportional to the pounds of carbon per square foot of hearth area per 24 hr., this method having been employed by other investigators.[22] After consulting all available data, I have come to the conclusion that in most cases there is but little economy in a rate of less than 4,000 lb. of carbon, and I have therefore taken a factor of 1.0 for this rate and under, while each increase of 100 lb. above 4,000 increases the factor by 0.01, so that for 5,000 lb. of carbon my factor is 1.1.

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[21] For a good illustration of this point see E. S. Cook, Anthracite and Coke, Separate and Mixed, in the Warwick Blast-Furnace, *Trans.*, xvii., 124 (1888–89).

[22] See more especially F. W. Gordon, *Trans.*, xx., 255 (1891), and M. A. Pavloff, The Rate of Combustion in Blast Furnaces, *Iron Age*, vol. lxxxiv, No. 9, p. 618 (Aug. 26, 1909).

4. *Loss of Carbon Between Throat and Hearth.*

In descending from the top to the hearth, carbon is lost in three ways: 1, through solution by the carbon dioxide of the flux; 2, through solution incidental to the reduction of the ore; 3, through solution in the pig-iron.

a. Solubility of Coke.—The first two of these losses are in some degree proportional to the quality of the fuel used. The fact that various forms of carbon and kinds of coke have a widely-differing degree of solubility in carbon dioxide has been abundantly proven by the laboratory-experiments of Boudouard,[23] Bell,[24] and probably others; and that there is a great difference in the actual fuel-economy given by various cokes is a matter of experience with every furnace-man.

While it is undoubtedly possible to find a quantitative relationship between the results of laboratory-experiments and practical value in the furnace, the necessary data are at present lacking and I have been forced to choose arbitrary values. I have assumed that for the best coke, such as that from the original Connellsville and Durham (England) fields, the factor of solubility will be 0.5, while for the worst cokes, as soft Pocahontas and the poorer Alabama varieties, it will be 1.0.

b. Carbon-Loss Due to Flux.—From the reactions involved it is evident that the maximum amount of carbon which can be dissolved by the carbon dioxide of the flux is 0.12 × the weight of $CaCO_3$ + 0.143 × the weight of $MgCO_3$, and it is usually considered that the actual loss of carbon due to this cause is very near the maximum. It is true that the substitution of crushed limestone for lump has resulted in a material saving of fuel, and this may indicate that some carbon dioxide escapes unchanged from the crushed stone. On the other hand, it may indicate that the large stone still retains some of its carbon dioxide when it reaches the hearth, since the heat absorbed in its decomposition there, being a direct charge upon *Ha*, would fully account for the greater fuel-consumption. In the absence

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[23] *Annales de Chimie et de Physique*, Series VII., vol. 24, pp. 5 to 85 (1901). Quoted in Dowson and Larter's *Producer Gas*, 1st ed. (1906).

[24] The Manufacture of Coke in the Hüssener Oven at the Clarence Iron Works and Its Value in the Blast-Furnaces, *Journal of the Iron and Steel Institute*, vol. lxv., p. 188 (No. I, 1904).

of quantitative data I have assumed that the factor representing the effect of size of stone on carbon-loss is 1.0 and 0.9 for lump and crushed stone respectively.

c. Effects of Reducibility of Ore.—The maximum amount of carbon which can be lost incidental to the reduction of the ore is the same whether the ore be reduced by carbon monoxide, but at such a high temperature that the resulting carbon dioxide at once dissolves its full quota of carbon, or whether it be reduced by solid carbon with the production of carbon monoxide. In either case this carbon-loss is 720 lb. per ton of iron, or about 700 lb. per ton of pig. The great desirability of having an ore which is readily reduced by carbon monoxide rather than by solid carbon, and in addition is reduced at such low temperatures that the resulting carbon dioxide has no solvent power, has been frequently pointed out. The importance of carbon-deposition in this connection does not, however, seem to be so generally appreciated. It will be recalled that this reaction, $2CO = CO_2 + C$, begins at about 430° and ceases entirely at 900°. That is, it takes place very near the top of the furnace. It is probable that very little of the carbon resulting from this reaction ever reaches the hearth, but it does useful work in reducing the carbon dioxide of the limestone and in removing that portion of the oxygen of the ore which has not been removed by carbon monoxide higher in the furnace. From this point of view it appears that the ability of an ore to induce carbon-deposition is equally as important as the ease with which it loses its oxygen.

It is, of course, true that an excessive deposition of carbon has its disadvantages, tending to increase the blast-pressure and cause hanging of the furnace; but granted that these objections can be overcome by suitable design and management of the furnace, it is certainly true that every pound of carbon deposited means a saving of a pound of fuel for the hearth. Incidentally, I would call attention to the fact that ores inducing large carbon-deposition should be particularly desirable in cases where it is necessary to use large percentages of limestone, and the greatest difficulties due to carbon-deposition are to be anticipated in cases where the limestone-requirements are very low.

Returning from this digression to the business of selecting

numerical factors of reducibility for the various classes of ores, I have selected more or less arbitrarily the following values:

Mesabi ores,	0.1
Brown hematite,	0.2
Soft red hematites and roasted carbonates,	0.3
Hard red hematites,	0.4
Clinton "hard red" ore (the limy ore of Alabama),	0.6
Magnetites and mill-cinders,	1.0

These values are in qualitative accordance with the results of laboratory-experiments,[25] and I believe that they also agree with the experience of most furnace-men. It is probably possible to construct a quantitative relationship between the results of laboratory-experiment on reducibility and practical results as regards fuel-economy, but I should hardly care to undertake the task at the present time.

5. *The Formula as Used.*

It will be evident from the foregoing discussion that although it is possible to construct with scientific accuracy the general expression for the fuel-requirements of a blast-furnace, when we come to apply this expression to practice we find there is either grave doubt or a complete lack of data as to practically every factor or constant entering into it. I feel, therefore, that I must apologize for having attempted the impossible, and my chief reason for having done so is to show the great value of this line of work in the study of the blast-furnace, and to emphasize more strongly the need of certain data in the hope that they may be more rapidly supplied.

The formula which I have actually used in the following calculations is as follows:

Carbon per ton of pig $= \frac{1{,}000}{Ha} \times (1{,}200 + 0.6 \times$ lb. of slag per ton of iron $+ 300 \times$ per cent. of Si) $\times$ factor of rate of

[25] See O. O. Laudig, Action of Blast-Furnace Gases upon Various Iron-Ores, *Trans.*, xxvi., 269 (1896), with discussion by F. E. Bachman, *Trans.*, xxvi, 1061. Also data in Bell's *Chemical Phenomena of Iron Smelting*, and statement by Gayley, *Trans.*, xix., 991 (1890–91). Some experiments on the relative reducibility of typical Alabama red ores, Lake Superior ores, and brown ores, have been made under my direction by W. J. Buvinger in the Metallurgical Laboratory of the University of Cincinnati, and these results also are in accordance with the values used.

driving + lb. of $Ca(Mg)CO_3 \times 0.12 \times$ size-factor of flux $\times$ quality-factor of fuel + $700 \times$ reducibility-factor of ore $\times$ quality-factor of fuel + $22.4 \times$ per cent. of carbon in pig.

This formula is of the same general construction as the expression previously given. All of the items coming under *Hn*, with the exception of heat to melt slag and heat to reduce silicon, have been lumped together in a single constant, which has been given the value of 1,200,000. The heat necessary to care for 1 lb. of slag has been taken at 600 B.t.u. This, it will be noted, is 50 per cent. higher than the most probable theoretical value, but in applying the formula it was found that better results were obtained from this larger figure if at the same time the constant was decreased correspondingly. I believe this to be due to the fact that my constant is not really constant, but increases with decreased output and therefore approximately with increased slag-volume.

III. The Limitations of the Formula.

1. *Through Irregular Working of the Furnace.*

I have aimed to be frank in admitting the crudity of this effort in the hope of forestalling criticism, but I am aware that upon one point it is particularly open to attack. It will at once occur to every practical furnace-man that no provision has here been made for the increase in fuel-consumption inevitably caused by irregularity in furnace-work; whether the latter be due to wrong furnace-lines, improper distribution of the charge, wear of lining, or other causes. This increase from irregular work is due in greater part to four items:

Channeling of the gases, resulting in increased velocity and decreased efficiency of heat-transmission.

Descent into the hearth of insufficiently preheated material, resulting in an increase in *Hn*.

Descent into the hearth of imperfectly reduced ore, thereby greatly increasing the amount of reduction to be performed in the hearth at the expense of *Ha*.

The descent into the hearth of undecomposed limestone, which by its decomposition absorbs 1,451 B.t.u. per pound of lime, this being a direct charge upon *Ha*.

I cannot see that it is possible to express numerically the re-

sults of furnace irregularity, and it seems that such a formula as I have proposed will, even when perfected, be limited to the itemizing of the fuel-requirements of the perfectly-working furnace.

2. *Through Ratio Between Heat-Requirements in Shaft and Hearth.*

The second limitation to my formula is because a certain quantity of heat and reducing gases are needed in the upper part of the furnace to do necessary work there. Ordinarily there is a large excess of both heat and carbon monoxide for this purpose, but it sometimes happens with furnaces smelting rich ores, especially if they use pure fuel and high values of. *Ha*, that the requirements here will be the limiting factor. This limitation has been discussed qualitatively from the standpoint of heat-requirements by Johnson,[26] and from the standpoint of carbon-requirements for reduction by Richards.[27] So far as I know, it has never before been treated from the quantitative stand-point.

The heat available to the stack, which we may designate by the symbol *Has*, is equal to the heat in the gases as they leave the hearth, or, assuming perfect heat-transfer between gases and solids, to the heat in the gases below the critical temperature. With average moisture in the blast and a critical temperature of 2,700°, $Has = 4{,}980 \times$ lb. of carbon burned in hearth.

The heat necessary in the stack, which we may designate by the symbol *Hns*, is as follows:

To preheat iron, approximately 1,200,000 B.t.u.

To preheat gangue, approximately 1,208 $\times$ lb. of slag per ton of iron.

To preheat carbon, approximately 1,133 $\times$ lb. of carbon burned in hearth per ton of iron.

To decompose the $CaCO_3$, 813 $\times$ lb. of $CaCO_3$ per ton of iron.

To compensate for loss by radiation, unknown.

To reduce iron, if by reaction $Fe_2O_3 + 3CO = 2Fe \quad 3CO_2$, there is an evolution of 281,230 B.t.u. If by reaction

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[26] *Trans.*, xxxvi., 483 (1906).

[27] Grüner's Ideal Working of a Blast Furnace, paper before the International Congress of Mining and Metallurgy, 1910. Reprinted in *Metallurgical and Chemical Engineering*, vol. viii., No. 7, p. 403 (July, 1910).

$2Fe_2O_3 + 3C = 4Fe + 3CO_2$, there are 878,472 B.t.u. required per ton of iron. If by reaction $Fe_2O_3 + 3C = 2Fe + 3CO$, 3,615,840 B.t.u. are required.

The heat carried out in the top-gases will vary with the temperature of the gases as well as with their weight. According to my views, the heat here is simply that left over from *Has* after the requirements of the stack are satisfied; that is, if *Has* is 3,000,000 B.t.u. and *Hns* is 2,000,000 B.t.u. per ton of iron, then the difference of 1,000,000 B.t.u., which cannot be otherwise used, must necessarily remain in the gases, causing them to pass out at some temperature depending on their weight. However, it is probably not fair to assume that the gases can pass out at much below 212° on account of the necessity of evaporating the moisture of the stock. Hence, we will include the heat carried out in the gases at 212° as a part of *Hns*. At 212° and an average specific heat of 0.25 the heat contained in these gases will be approximately 6.7 × 53 × lb. of carbon burned in hearth per ton of iron + 900 × 53 + 0.44 × lb. of limestone per ton of iron × 53 + 0.12 × lb. of limestone per ton of iron × 53, which reduces to 47,700 + 355 × lb. of carbon burned in hearth per ton of iron + 30 × lb. of flux per ton of iron.

It is probably fair to assume that in most cases the heat absorbed through solution of carbon by carbon dioxide is just about balanced by the heat evolved in the deposition of carbon. Granting this and collecting all the above items, we have: *Hns* = 966,470 + 1,488 × lb. of carbon burned in hearth per ton of iron + 1,208 lb. of slag per ton of iron + 843 × lb. of flux per ton of iron, if the iron is all reduced by carbon monoxide. If the iron is reduced by solid carbon, the constant in this expression is increased, the other terms remaining the same. Numerically, the value for the first term becomes 2,126,172 B.t.u. and 4,863,540 B.t.u. per ton of iron when the solid carbon is oxidized to CO_2 and CO respectively.

To simplify this expression further, let us apply it to a specific type of furnace-practice; that is, Northern practice using Lake Superior ores. In this case we may assume that 90 per cent. of the ore is reduced by carbon monoxide and 10 per cent. by solid carbon with the formation of carbon monoxide. We will also assume that the weight of slag per ton of iron

is 0.715 times the weight of limestone, and we will ignore the loss of heat through conduction and radiation. Under these circumstances *Hns* reduces to 1,356,180 + 1,488 $\times$ lb. of carbon in hearth per ton of iron + 1,810 $\times$ lb. of slag per ton of iron.

Evidently, if *Ha* and not *Has* is to be the limiting factor, *Has* must be equal to or greater than *Hns*, or pounds of carbon in hearth per ton of iron must be equal to or greater than 400 + 0.52 $\times$ lb. of slag per ton of iron, and, since the pounds of carbon in the hearth equals *Hn* ÷ *Ha*, we have as the final expression

$$\frac{Hn}{400 + 0.52 \times \text{lb. of slag per ton of iron}} \text{ must be equal to or}$$

greater than *Ha* if the fuel-requirements of the blast-furnace are to be determined by the heat necessary and available in the hearth.

Although the above expression can make no great claims to accuracy, it is believed that calculations along this line can be made with sufficient exactness to be of service. Their value should be very great in certain cases, as, for example, in determining whether efforts to increase *Ha* are worth while in any given case. To illustrate, assume a furnace working on 1,000 lb. of slag per ton of pig-iron, making iron with 1 per cent. of silicon and with *Ha* = 1,500 B.t.u. Applying our formula, we find that the highest value of *Ha* which can be of use in decreasing fuel-consumption is 2,280. Evidently in this case there is considerable margin for improvement, but if *Ha* had already been in the neighborhood of 2,000 it is just as evident that it would be money wasted to install dry-blast plant or better stoves unless at the same time *Hns* were increased by calcining the limestone, as suggested by Johnson,[28] or by other means.

Another phase of this subject of the limitation of fuel-economy is found in the consideration of the quantity of carbon required for reduction. To reduce 1 ton of pig-iron by means of carbon monoxide requires, according to the reaction, 700 lb. of carbon, while to reduce the same iron by means of solid carbon requires either 700 or 350 lb. of carbon, according as it is oxidized to CO or CO_2.

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[28] *Trans.*, xxxvi., 486 (1906).

Richards, in the paper previously quoted, assumes that the reactions of reduction are as follows:

$$Fe_2O_3 + 6CO = 2Fe + 3CO_2 + 3CO, \text{ and}$$
$$Fe_2O_3 + 2C = 2Fe + CO_2 + CO$$

requiring 1,400 lb. of carbon per ton of pig-iron in the first case and 467 lb. in the second case. From this he draws the conclusion that when the heat-requirements of the hearth are small it is best to have reduction by means of solid carbon.

However, since it is not possible to make pig-iron with a slag-volume of much less than 800 lb. per ton of iron, it would seem that the heat-requirements of the shaft will, at least in coke-furnace practice, prevent any material economy through reduction by solid carbon. In the case of a slag-volume of 800 lb. per ton of iron the carbon burned in the hearth cannot be less than 816 lb. per ton of iron and must probably be considerably more to satisfy *Hns*. This is assuming that 90 per cent. of the reduction is by carbon monoxide, in the top of the furnace. If the reduction is more largely by solid carbon, as Richards claims, the heat-requirements in the shaft of the furnace will be very much increased, and the necessarily greater amount of fuel will at once insure a larger amount of carbon monoxide in the top gases.

I am strongly inclined to hold with Johnson [29] that the carbon-ratio of the gases is an effect, not a cause, and that the real limit to fuel-economy lies in the heat-requirements of the various zones of the furnace. At the same time, it is probably true that the place and method of reduction are influenced to a considerable extent by the relation between *Hn* and *Hns*.

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[29] *Trans*, xxxvi., 485 (1906).

IV. THE FORMULA APPLIED TO ACTUAL FURNACE-RECORDS.

TABLE I.—*Description of Furnace-Stock.*

No.[a]	Kind of Ore.	Ore-Factor	Kind of Fuel	Fuel-Factor	Lime-stone-Factor.
1	L Sup. ½ Mesabi	0.1	Lower Connellsville	0.6	0.9
2	L Sup. Old Range	0.3	Connellsville?	0.6	0.9
3	L. Sup. ½ Mesabi	0.1	Connellsville	0.6	0.9
4	L. Sup. ½ Mesabi	0.1	Connellsville	0.6	0.9
5	L. Sup. 80 per cent Mesabi	0.1	Connellsville & Pocah	0.7	0.9
6	L Sup. 80 per cent. Mesabi	0.1	Connellsville & Pocah.	0.7	0.9
7	L. Sup. Old Range.	0.3	Connellsville	0.5	0.9
8	L Sup Old Range	0.3	Lower Connellsville?	0.7	0.9
9	L. Sup Old Range	0.3	By-product Coke	0.5	0.9
10	L. Sup. Mostly Mesabi	0.1	Lower Connellsville?	0.6	0.9
11	L. Sup. Mostly Mesabi	0.1	Lower Connellsville?	0.6	0.9
12	L. Sup. Mostly Mesabi	0.1	Lower Connellsville?	0.6	0.9
13	L. Sup. ½ Mesabi	0.1	Lower Connellsville?	0.6	0.9
14	L. Sup Mostly Mesabi	0.1	Lower Connellsville?	0.6	0.9
15	L. Sup. Mostly Mesabi	0.1	Lower Connellsville?	0.6	0.9
16	L. Sup. Old Range	0.4	Connellsville	0.5	0.9
17	L. Sup. Old Range	0.4	Connellsville	0.5	0.9
18	L. Sup. Old Range	0.4	Connellsville	0.5	0.9
19	L. Sup. 70 per cent. Mesabi	0.1	Connellsville?	0.5	0.9
20	L. Sup. Some Mesabi	0.2	Connellsville?	0.5	0.9
21	Magnetic Concentrates	1.0	Anthracite & Coke?	0.7	0.9
22	Magnetic Concentrates	1.0	Anthracite & Coke?	0.7	0.9
23	Magnetic Concentrates	1.0	Anthracite & Coke?	0.7	0.9
24	Roasted Carbonate	0.3	Durham, Eng., Coke	0.5	0.9
25	? German Ores	0.4	? German Coke	0.8	0.9
26	? German Ores	0.2	? German Coke	0.8	0.9
27	? German Ores	0.4	? German Coke	0.8	0.9
28	? French Ores	0.2	?	0.8	0.9
29	Oriskany Brown Ore, Va.	0.2	Soft Pocah. Coke	1.0	1.0
30	Oriskany Brown Ore, Va.	0.2	Pocah. Coke	0.7	1.0
31	Oriskany Brown Ore, Va.	0.2	New River Coke	0.7	1.0
32	Oriskany Brown Ore, Va.	0.2	New River Coke	0.7	1.0
33	Alabama Brown Ore	0.2	Alabama Soft Coke	0.8	1.0
34	0.8 Brown, 0.2 Soft Red, Ala.	0.4	Alabama Soft Coke	0.8	1.0
35	0.7 Brown, 0.3 Soft Red, Ala.	0.4	Alabama Soft Coke	0.8	1.0
36	0.9 Brown, 0.1 Soft Red, Ala.	0.3	Alabama Soft Coke	0.8	1.0
37	0.8 Ala. Hard Red Ore	0.6	Very Soft Coke	1.0	1.0
38	0.8 Ala. Hard Red Ore	0.6	Very Soft Coke	1.0	1.0
39	0.8 Ala. Hard Red Ore	0.6	Very Soft Coke	1.0	1.0
40	0.8 Ala. Hard Red Ore	0.6	Very Soft Coke	1.0	1.0
41	0.8 Ala. Hard Red Ore	0.6	Very Soft Coke	1.0	1.0
42	0.8 Hard Red Ore, Ala	0.6	Very Soft Coke	1.0	1.0
43	⅔ Hard, ⅓ Soft Red Ore	0.6	Ala. Coke, Pratt Seam	0.8	1.0

[a] For references, see p. 212.

TABLE II.—*Furnace Data.*

Number [a]	Temperature of Blast	Grains of Water Per Cu. Ft of Blast.	Lb. of Slag Per Ton of Iron.	Per Cent. Silicon in Pig	Lb. Ca(Mg)CO_3 Per Ton Iron	Per Cent Fixed Carbon in Coke.	Lb. Coke Per Ton Iron	Lb. of Carbon Per Sq. Ft of Hearth-Area Per 24 Hr.
1	720	5.66	1,300	0.8	945	88.0	2,147	4,265
2	1,040	4.00	1,388	2.1	1,318	79.0	2,258	4,601
3	915	4.00	1,300	1.3	1,102	88.0	1,936	4,330
4	885	4.00	1,350	1.1	1,091	88.0	2,217	5,360
5	700	4.00	1,300	1.5	1,084	88.0	2,490	4,800
6	750	4.00	1,050	1.1	837	88 0	2,180	3,770
7	1,100	4 00	1,200	1.4	1,011	88.0	1,882	4,600
8	1,027	6.50	900	1.1	783	88.0	1,955	4,000
9	925	6.50	820	1.1	700	85.0	1,931	4,100
10	807	3 92	1,380	0.8	1,077	88.0	2,275	5,000
11	863	3.50	1,260	0.8	965	88.0	2,256	5,300
12	775	3.30	1,530	0.8	1,232	88.0	2,258	5,000
13	870	1.75.	1,250	0.8	945	88.0	1,726	4,118
14	854	1.30	1,350	0.8	1,052	88.0	1,821	4,880
15	820	1.20	1,330	1.2	1,030	88.0	1,961	5,000
16	1,130	4.00	900	1.5	800	88.0	1,580	3,600
17	1,202	4.00	672	1.4	605	88.0	1,630	3,270
18	1,058	4.00	672	1.4	582	88.0	1,900	3,200
19	1,100	1.90	1,100	0.7	878	88.0	1,977	4,600
20	1,108	3.00	950	1.2	771	83 1	1,664	3,350
21	900	3.00	1,200	0.9	1,036	88.0	2,291	4,000
22	900	3 00	1,400	0.9	1,267	88.0	2,437	4,000
23	900	3.00	1,200	1.3	1,081	88.0	2,241	4,000
24	1,300	4.00	3,136	3.0	1,232	92 0	2,239	3,000
25	1,400	4.00	1,800	3.0	1,500	85.0	2,395	3,000
26	932	4.00	2,500	1.0	1,500	90.0	2,128	3,000
27	1,382	4 00	3,500	2.0	2,800	90.0	2,740	3,000
28	1,250	4.00	2,500	1.0	1,500	80.0	2,120	3,000
29	1,000	6.00	3,400	0.8	3,270	92.0	3,370	3,870
30	1,000	5.00	2,700	2 4	2,340	90.0	3,132	4,250
31	1,100	6.00	3,100	2.2	2,800	88.0	3,260	3,200
32	1,100	6.00	3,300	2.2	3,000	88.0	3,290	3,200
33	950	6.00	3,300	2.5	3,051	80.0	3,835	3,600
34	1,000	4.00	2,700	2 5	2,400	85.0	3,283	4,000
35	1,075	4.00	2,800	2.3	2,500	85.0	3,267	4,000
36	1,000	5.00	2,700	2.5	1,740	85.0	3,302	4,000
37	900	8.00	2,130	2.2	2,050	91.0	3,840	5,430
38	1,000	6.00	2,200	2.0	2,000	91.0	3,380	4,850
39	1,000	8.00	2,170	2.3	1,890	91.0	3,354	4,760
40	1,000	6.00	2,300	2.2	2,100	91.0	3,240	5,000
41	1,000	8 00	2,230	2.5	1,930	91.0	3,792	4,780
42	1,000	8.00	2,130	2.2	1,910	91.0	3,607	5,180
43	1,400	6.00	2,180	2.0	1,850	86.7	2,240	3,700

[a] For references, see p. 212.

TABLE III.—*Application of Formula.*

Number.[a]	Heat Available Per Lb Carbon (Ha).	Heat Necessary Per Ton Iron ÷ 1000 $\left(\frac{Hn}{1000}\right)$	Factor of Rate of Driving	Lb Carbon in Hearth.	Carbon-Loss in Shaft.	Calculated Carbon Per Ton Iron	Actual Carbon Per Ton Iron	Error.
1	1,225	2,220	1.03	1,866	181	2,047	1,889	+ 158
2	1,775	2,663	1.06	1,595	290	1,885	1,783	+ 102
3	1,600	2,361	1.03	1,520	191	1,711	1,702	+ 9
4	1,550	2,340	1.14	1,690	191	1,881	1,950	— 69
5	1,290	2,418	1.00	1,875	209	2,084	2,146	— 62
6	1,365	2,160	1.01	1,602	191	1,793	1,918	— 125
7	1,870	2,340	1.06	1,326	238	1,564	1,620	— 56
8	1,600	2,070	1 00	1,293	285	1,578	1,720	— 142
9	1,450	2,010	1.01	1,400	221	1,621	1,641	— 20
10	1,450	2,328	1.10	1,768	190	1,958	2,000	— 42
11	1,550	2,196	1.13	1,599	183	1,782	1,980	— 198
12	1,430	2,358	1.10	1,815	200	2,015	1,980	+ 35
13	1,650	2,190	1.01	1,338	181	1,519	1,519	0
14	1,650	2,250	1.09	1,489	188	1,677	1,600	+ 77
15	1,610	2,350	1.10	1,606	187	1,793	1,734	+ 59
16	1,920	2,190	1.00	1,142	261	1,403	1,390	+ 13
17	2,025	2,053	1.00	912	250	1,162	1,434	— 272
18	1,810	2,053	1.00	1,134	249	1,383	1,669	— 286
19	2,000	2,085	1.06	1,106	160	1,266	1,739	— 468
20	1,935	2,049	1.00	1,058	190	1,248	1,403	— 155
21	1,625	2,190	1.00	1,345	646	1,991	2,015	— 24
22	1,625	2,310	1.00	1,420	664	2,084	2,145	— 61
23	1,665	2,310	1.00	1,420	650	2,070	1,970	+ 100
24	2,175	3,992	1.00	1,836	250	2,086	2,060	+ 26
25	2,325	3.180	1.00	1,370	431	1,801	2,035	— 243
26	1,625	3,000	1.00	1,845	319	2,164	1,915	+ 249
27	2,290	3,900	1 00	1,705	544	2,249	2,466	— 217
28	2,100	3,000	1.00	1,430	319	1,748	1,696	+ 54
29	1,525	3,480	1.00	2,280	622	2,902	2,880	+ 22
30	1,585	3,540	1.02	2,305	384	2,689	2,679	+ 10
31	1,675	3,720	1.00	2,221	423	2,644	2,727	— 83
32	1,675	3,840	1.00	2,292	440	2,732	2,755	— 23
33	1,525	3,930	1.00	2,574	505	3,079	3,070	+ 9
34	1,725	3,570	1.00	1,974	544	2,518	2,651	— 133
35	1,830	3,510	1.00	1,912	554	2,466	2,638	— 172
36	1,660	3,570	1.00	2,149	488	2,637	2,665	— 28
37	1,340	3,129	1.14	2,660	744	3,404	3,494	— 90
38	1,590	3,108	1.08	2,211	738	2,949	3,075	— 126
39	1,480	3,177	1.08	2,317	719	3,036	3,042	— 8
40	1,590	3,243	1.10	2,244	750	2,994	2,951	+ 43
41	1,480	3,294	1.08	2,403	730	3,133	3,450	— 317
42	1,480	3,150	1.12	2,386	727	3,113	3,282	— 169
43	2,200	3,108	1.00	1,414	592	2,006	1,940	+ 66

a For references, see p. 212.

Foot-note References to Tables I., II., III., and IV.

No.	Reference.
1.	*Trans.*, xxxv., 762; xxxvi., 746; xl., 919.
2.	*Trans.*, xl., 919.
3, 4.	*Trans.*, xxxix., 545.
5, 6.	Private notes.
7.	*Manufacture and Properties of Iron and Steel*, by H. H. Campbell, 2d ed., p. 76.
8, 9.	*Iron Age*, June 11, 1903, p. 33.
10, 11, 12.	*Trans.*, xxxvii., 206.
13.	*Trans.*, xxxv., 762; xxxvi, 746; xl., 919.
14, 15.	*Trans.*, xxxvii., 206.
16.	*Trans*, xxxix., 906.
17, 18.	*Trans.*, xx., 287.
19.	*Metallurgical and Chemical Engineering*, June, 1910, p. 315.
20.	*Trans.*, xxvii., 477.
21, 22, 23.	*Iron Age*, May 6, 1909, p. 1438.
24.	*Manufacture and Properties of Iron and Steel*, by H. H. Campbell, p. 76.
25.	*Trans.*, v., 330.
26, 27.	*Trans.*, xix., 346.
28.	*Trans.*, xxxv., 1039.
29 to 42.	Author's private notes.
43.	*Trans.*, xvii., 135.

TABLE IV.—*Average Furnace-Practice.*

Numbers[a]	1–12.	13–15	16–20	21–23	24–28.	29–36.	37–43.
Furnace Practice	L. Sup. Ore Natural Blast.	L Sup Ore Dry Blast	L Sup Ore High *Ha.*	Magnetic Ore	Foreign.	Southern Brown Ore.	Ala Red Ores
Number of Records Averaged.	12.	3.	5	3	5.	8	7
Temperature of blast, degrees Fahrenheit	875	850	1,120	900	1,250	1,030	1,040
Water per cubic foot of blast, grains	4 45	1 41	3 50	3 00	4.00	5 25	7 10
Average critical temperature, estimated.	2,700	2,700	2,700	2,700	2,700	2,725	2,700
Slag per ton of pig-iron, pounds	1,120	1,310	800	1,270	2,690	3,000	2,200
Silicon in pig-iron, per cent.	1.15	0 90	1 25	1 05	2 00	2.20	2 20
Carbon in pig-iron, per cent.	3 50	3 50	3 50	3 50	3.50	4 00	3 50
Carbonates per ton of pig-iron, pounds	1,010	1,010	725	1,130	1,710	2,720	1,960
Carbon per square foot of hearth per 24 hr., pounds	4,600	4,600	3,600	4,000	3,000	3,800	4,800
Fixed carbon in coke, per cent	88 0	88 0	88 0	88 0	87.0	88.0	90 0
Kind of coke	Mostly Connellsville	Mostly Connellsville	Mostly Connellsville	Anthracite and Mt Coke	Various.	W Va. and Ala.	Ala Soft
Coke per ton of pig-iron, pounds	2,150	1,836	1,740	2,323	2,344	3,342	3,350
Coke-factor	0.6	0 6	0 5	0 7	0 7	0 8	0 9
Stone-factor	0 9	0.9	0 9	0 9	0 9	1 0	1 0
Reducibility of ore	0 2	0 1	0 3	1.0	0 3	0 2	0 6
Heat available (*Ha*)	1,510	1,650	1,945	1,630	2,100	1,690	1,510
Heat necessary (*Hn*)	2,217,000	2,256,000	2,091,000	2,277,000	3,414,000	3,660,000	3,180,000
Factor of rate of driving	1.06	1.06	1 00	1 00	1 00	1 00	1 08
Carbon in hearth	1,555	1,449	1,085	1,400	1,625	2 170	2,160
Carbon-loss	228	185	220	653	354	463	668
Theoretical total carbon	1,783	1,634	1,290	2,053	1,979	2,633	2,828
Actual total carbon	1,892	1,615	1,608	2,014	2,039	2,794	3,015
Error	— 109	+ 19	— 318	— 9	— 60	— 161	— 187

[a] For references, see p. 212.

V. Sources of Error.

Although many and wide variations from the fuel-requirements called for by theory will be noted in Tables I. to IV., it is thought that the results are, on the whole, fairly satisfactory, and indicate something more than coincidence. The errors referred to are no doubt partly due to limitations in the formula which have been previously discussed. Many of the furnaces included are known to have been subject to irregularities from one source or another, and these are probably accountable in some instances for the considerable excess of actual fuel above that called for by theory.

Attention is particularly called to the group of furnaces included between Nos. 16 and 20. It will be noted that these furnaces all show extremely low fuel-requirements in the hearth, and that, with one exception, the calculated fuel-consumption is considerably below the actual. I believe that in some, if not all, of these cases, the second limitation previously described, that is, the heat-requirements of the stack, has come into play, and that this is in part, at least, the reason for the difference. Other sources of discrepancy may be sought in errors in the records before attributing them entirely to the imperfection of the formula.

The temperature of the hot blast is one of the most important factors entering into our calculation, and an error of 50° here may in some cases make a difference of as much as 100 lb. in the total carbon required per ton of iron. It is also a record which is particularly liable to error, both because of the difficulty of obtaining a true average when it is observed intermittently and because of the common inaccuracy of the pyrometers used. The older forms of pyrometers were particularly liable to high readings, and old furnace-records must be used cautiously for this reason.

The moisture of the blast is another factor which is undoubtedly in error in many cases, since, for the most part, no records were available, and all that could be done was to make a guess based upon the location and the season of the year. In this connection, the possibility of additional water entering the furnace through small and perhaps unsuspected cracks in tuyères or hot-blast valves may well be considered.

The critical temperature is the most uncertain of all our fac-

tors. Since it is only possible to guess it, a uniform value of 2,700° F. was taken in all cases, except with certain Virginia furnaces which were known to run on a slag unusually low in alumina, and hence of considerably higher melting-point than the average. The critical temperature here was taken as 2,750°. It is probable that there is considerable variation in this factor, especially among those foreign furnaces which run on a very aluminous slag. A difference of 100° in critical temperature makes a difference of 150 B.t.u. in heat available.

Other possible errors are in slag-volume, which in most cases has been calculated from the quantity of flux used, and in some cases from general data on the yield and nature of the ores; in the percentage of silicon, which it has been necessary to estimate in some cases; and in the weight of flux and coke. In connection with this latter the practice of forking-out the breeze at some furnaces should be borne in mind. An approximate correction has been made for this coke not going into the furnace in several cases where the practice was known to be followed, but there are, no doubt, other cases in which it constitutes a source of error.

VI. Suggestions for Practical Methods of Fuel-Economy.

It is evident from this analysis of fuel-requirements that it is in connection with the heat available that the greatest opportunities for the saving of fuel are found. A long step in this direction was taken when Neilson invented the hot blast, and another, hardly less important, when Gayley demonstrated the value of dry blast. Until commercially-feasible methods of producing high-oxygen blast are at hand it would seem that there is no new ground to be broken (so far as *Ha* is concerned) and that the activities of furnace-managers are limited to the more perfect cultivation of the fields already open.

In the first three groups of the preceding tables the importance of both high blast-heat and dry blast are very evident. In connection with the close agreement of the dry-blast furnaces with theory and the considerable minus error of the other two groups, it should be remembered that one effect of dry blast is to give great regularity in operation, and thus, by decreasing unnecessary fuel-losses, it becomes a double benefit.

It has already been pointed out that in the case of the third

group, the high average minus error is possibly due to the very low heat-requirements of the hearth being surpassed by the requirements of the stack. I wish, however, to now make the point that this condition is at present less frequent than in previous years, and will become still less frequent in the future. The lowering of the average grade of our iron-ores has been frequently commented upon, and there can be no question but that the furnace-men of the future will have to meet the condition of smelting very lean ores. With the high slag-volume thus produced, the carbon-requirements of the hearth will become more than ever the controlling factor, and the heat-requirements in the stack may, for all practical purposes, be ignored.

An example of what may be expected under these conditions is found in our fifth group, representing foreign practice. These five records are among the least satisfactory in respect to accuracy of the data, but they will at least serve to indicate the very low fuel-consumption which can be attained even in the case of very lean ores by the use of high blast-temperatures. The average heat available for this group is 2,100 B.t.u. per pound of carbon, and it is obtained by an average blast-temperature of 1,250°.

As compared with this, recent American practice, represented by groups 1, 2, 4, 6, and 7, shows average heats available of 1,510, 1,650, 1,630, 1,690, and 1,510 B.t.u., respectively, and the average for blast-temperatures ranges from 850° to 1,040°. With these figures in view we can hardly avoid the suspicion that we in this country are perhaps in a rut with respect to the proper use of blast-heat, and are not devoting the attention to this factor which its importance deserves.

It is true that when using Mesabi ores the blast-temperature is limited by the tendency of the furnace to stick and hang at high heats. That this difficulty is not insurmountable, however, is indicated by the results obtained by some furnace-managers, such, for example, as are shown in No. 19, in which 1,100° of blast-heat is used in connection with 70 per cent. of Mesabi ore.

There is, of course, a limit above which it is neither practicable nor especially desirable to carry the available heat. This follows not so much from the limitations of our hot-blast stoves as from the fact that fuel-economy is proportional not to *Ha*

directly, but to its reciprocal, which decreases at a decreasing rate as *Ha* becomes larger. This fact, together with its attendant results on fuel-consumption, is shown graphically in Fig. 3. It is at once evident that with high values of *Ha* only

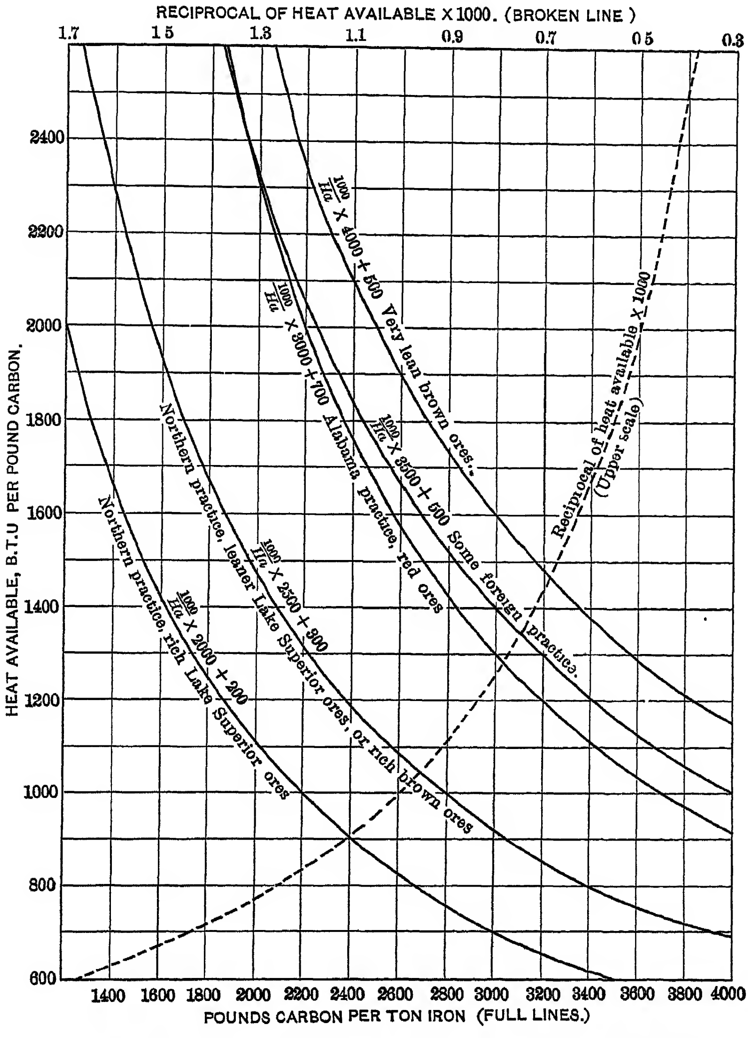

FIG. 3.—RELATION BETWEEN AVAILABLE HEAT AND FUEL-CONSUMPTION.

a small economy in fuel is effected by further increase, while with low values the economy for a given increase is very much greater. This explains the relatively great saving produced by

the first few hundred degrees of blast-heat and the relatively small additional benefits derived from very high temperatures. These facts are of common knowledge and recently have been well presented in graphical form by Moore.[30]

Fig. 3 shows also that the fuel-economy for a given increase in *Ha* is greater in the case of lean ores having a high heat-requirement than with rich ores. Efforts to increase *Ha* will, therefore, pay better in the case of Alabama and brown ore practice than where the rich Lake Superior ores are being used. In this connection it is also true that dry blast will be especially advantageous at these Southern furnaces. By reference to the tables it will be noted that the average humidity is very high, and hence the available heat is quite low, in spite of the fairly high blast-temperature.

Since the heat necessary in the hearth is fixed chiefly by commercial factors beyond the control of the furnace-manager it is not usually possible to do much towards reducing this item. I would, however, point out the possibility of predicting through a study of this factor the quantitative effect of a new ore or ore-mixture on fuel-economy, or the saving to be expected through washing or other concentration of the ore.

In respect to carbon-loss in the shaft of the furnace, it is thought that current furnace-practice is, as a rule, fairly satisfactory with the exception that in the Southern districts the ore and limestone are generally insufficiently crushed and sized previous to charging. I believe that in a number of cases it would be possible to save at least 100 lb. of fuel per ton of iron by better practice in this regard. In the case of easily-reducible ores, such as the Mesabis, fineness beyond a certain point is undesirable, since it induces an excessive carbon-deposition and causes hanging of the furnace. On the other hand, difficultly-reducible ores do not cause this trouble, and fuel may be saved by having them as finely crushed as is compatible with a reasonably low loss in flue-dust. This statement is abundantly confirmed by experience with magnetic concentrates.

In conclusion, I hope that this paper may do something

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[30] The Fuel Economy of Dry-Blast as Indicated by Calculations from Empirical Data, *Journal of the Iron and Steel Institute*, vol. lxxx., p. 150 (No. II., 1909).

towards arousing interest in the more exact study of blast-furnace operations, and, in particular, may lead to the publication of more complete and accurate data by those actively engaged in furnace management. It is freely admitted that the details in the method of calculation here proposed may need revision, but it is only through the accession of more accurate data that the needful corrections can be made.

VII. Efficiency as a Factor in Furnace Management.

1. *The Efficiency Principle.*

Efficiency of management may be defined as the ratio of the actual results to the best possible theoretical results.

Simple as this proposition may appear, it is far from being universally accepted as the proper guage or standard of management, and it is only within recent years and in a limited number of industries that the principle has been applied at all. The difficulty, of course, is in the determination of the best possible theoretical results, or, in other words, in the setting of the standard for comparison. The study of manufacturing operations with a view to obtaining this information and standardizing both results and methods of obtaining them is, under the name of production engineering, now receiving recognition as a special branch of engineering science.

From this stand-point manufacturing operations may be divided into two groups: 1, those in which labor is the chief item of cost; and 2, those in which the material cost offers the chief opportunity for saving.

The best example in the first group is the machine-shop, which industry has been particularly benefited by the work of F. W. Taylor, the pioneer in production engineering. Taylor's problem here was two-fold: 1, the theoretical maximum output of each machine must be found; 2, it was necessary to find means of persuading their human operators to closely approach this maximum. His solution was reached through careful time-study of the elements of every operation, combined with the use of various task-systems as a basis for the payment of labor. The successful issue of his studies is best appreciated through a perusal of his monumental work, Shop Management.[31]

[31] *Transactions of the American Society of Mechanical Engineers*, vol. xxiv., pp. 13·7 to 1456 (1903).

An excellent example of the application of efficiency methods to an industry of the second type is found in the work of Charles Catlett on the beehive coking process.[32] In this case, through the establishment of a system of daily efficiency-records, both output and yield of coke were increased to a remarkable degree, costs being correspondingly lowered.

In my opinion this work has never received the attention which it deserves. Its importance is due not only to the results accomplished, which are a striking illustration of the value of efficiency records, but also to its perfect application of the general principle involved.

In addition to these two industries, the efficiency principle has been applied in a number of instances to foundries, the work of Harrington Emerson and associates being especially noteworthy in this connection. Recently I have become cognizant of efforts which are being made to apply it to mining-operations.

2. *The Application to the Blast-Furnace.*

About five years ago the work of Taylor and Catlett came to my attention, and through interest in their results I was led to examine the feasibility of applying the same principles to blast-furnace operations. So far as known, there has thus far been no systematic effort looking towards the use of this idea in the manufacture of iron, and it is hoped that this investigation may be a definite step in that direction.

The first question involved in this application is the qualitative definition of "best results." In other words, what is the object for which the blast-furnace is operated? The answer to this, of course, is profits. Carrying the analysis still further, we find a large number of items upon which efficiency depends, but the four which are of chief importance in the technical operation of the furnace are, output, quality, fuel-consumption, and labor-cost.

The especial importance of fuel-consumption follows not only because it is a factor in the determination of both output and labor-cost, but also because it is the largest single item in the cost-sheet which is subject to the control of the furnace-manager. This is well shown by Table V.

[32] Coking in Beehive Ovens with Reference to Yield, *Trans.*, xxxiii., 272 (1903).

TABLE V.—*Approximate Percentage of the Items Entering into the Cost of Making Pig-Iron.*

	Alabama	Virginia.	Pittsburg.	Chicago.	Atlantic Coast.
	Per Cent.	Per Cent.	Per Cent.	Per Cent	Per Cent.
Ore	35	38	70	54	43
Flux	1	7	2	2	5
Fuel	47	39	18	34	40
Labor	10	11	6	6	8
Supplies and repairs	7	5	4	4	4
	100	100	100	100	100

Tremendous advances have been made within recent decades in the development of labor-saving machinery designed for the purpose of reducing the relatively unimportant item of labor-cost. I would not question the value of such improvements as skip-hoists, casting-machines, pig-breakers, etc. I believe, however, that a simultaneous study of the possibility of metallurgical improvement and of the cost-sheet will show that in many cases far greater returns could have been obtained through judicious expenditure looking towards the reduction of fuel through an increase in available heat.

Recalling my own experiences, I am inclined to think that most furnace-men place too much emphasis upon the mechanical equivalents of labor, having in mind their difficulties in the supply of labor, and losing sight of the relatively unimportant effect of these improvements upon the cost-sheet.

In applying the efficiency principle to the fuel-requirements of the blast-furnace we are immediately confronted with the lack of a proper basis for the analysis of the fuel-requirements and the calculation of the theoretical best, or standard performance. It was to supply this need that the present investigation was undertaken, and it is hoped to supplement it in the future by additional papers on the other factors of efficiency.

The United States Iron Industry from 1871 to 1910.

BY JOHN BIRKINBINE, PHILADELPHIA, PA

(Wilkes-Barre Meeting, June, 1911)

MODERN advances in practically all lines of industrial development have occurred in such rapid succession, and have been accepted so readily as accomplished facts, that a retrospect surprises us, by showing how comparatively few of the acknowledged factors of improved conditions may be considered as old. While these advances have not been confined to any country, they have been more pronounced in some than in others, and nowhere more so than in the United States, the population of which, having multiplied nearly three-fold between 1870 and 1910, demanded a proportionately greater increase in materials, supplies, and manufactured products. It therefore appears desirable to discuss mainly conditions in the United States, as concrete evidence of industrial progress throughout the world.

Looking backward for but three generations, we may trace the introduction and development of canal- and steamboat-navigation; railroad-transportation; artificial illumination beyond that furnished by candles and animal-oils; quick communication by mail, and subsequently by telegraph or telephone; the manufacture of iron, beyond forms of small dimensions; the production and utilization of steel in large quantities; the economic use of mineral fuel, oil and gas, etc.

The practical coincidence of the fortieth anniversary and the 100th technical meeting of the American Institute of Mining Engineers, offers temptation to recall and compare the conditions of mining and metallurgy in or about the years 1871 and 1911.

A complete *résumé* would cover phenomenal changes in mining-methods and equipment by which the output of individual exploitations has grown from scores to hundreds and even thousands of units in equal time-intervals. Extension of operations in depth and area, demanding machinery of great power and efficiency, high percentage of extraction, utilization of what

was formerly waste, and the beneficiation of inferior mineral; the employment of mechanical appliances in exploration and development, and of vehicles for transportation which take care of large quantities at low cost per ton-mile; and improved methods of mine-working and mine-supports—are among the factors of mining progress. The appliances by means of which, in four decades, the annual coal-production of the United States was increased 14-fold, and millions of tons of material which, in 1871, passed to the waste-piles were industrially utilized, are merely instances of this progress.

An equal advance in metallurgy has been effected by the combined efforts of the chemist, the metallurgist, the mechanical engineer, and, lately, the electrical engineer.

Of the advances made during the life of the Institute, the record of the pig-iron industry is presented as a typical example; for on this industry is based the marvelous development of the American steel manufacture, and of the industries employing steel as a material.

The relatively insignificant production of Bessemer steel, reckoned in "long" tons of 2,240 lb. av. (about 35,700 in 1870), grew to 9,500,000 tons in 1910, and the product of open-hearth steel, of less than 1,400 tons in 1870, has increased until last year 16,500,000 tons were made in the United States.

The railroad-mileage of the country (60,000 miles in 1870) has been augmented to 343,000 miles; and the construction of more than 40,000 miles of trolley-systems, together with the introduction of steel structural work, has been responsible for much of the increased consumption of steel.

The history of the Institute covers the introduction of natural gas in the manufacture of iron and steel; the predominant employment of coke as a fuel for blast-furnaces; the production of basic steel, and the general replacement of iron by steel; the use of mixers for molten metal; the manufacture of American tin-plate; the construction of iron or steel vessels, armor-plate, steel cars, and structural steel buildings; the installation of pipe-lines to convey oil; the utilization of electricity for light and power, and the creation of great industries for the production of cement, aluminum, and metallic alloys, and for the manufacture of bicycles, automobiles, and the apparatus of aviation.

The application of high-pressure steam or air, of water-turbines under high heads, of high-speed machinery and tools, of high-temperature blast, and of high explosives, are features of the four decades of the Institute's existence; as are also the advances in practical electro-metallurgical processes, and many instances of utilization of by-products or waste material.

In iron-smelting, regenerative hot-blast stoves, by-product coke- and charcoal-ovens, skip-hoists, liberal water-cooling, gas-engines, and the dry-air blast, are special developments of the same period, contributing to increase of product, decrease of fuel-consumption, reduction of labor-cost, and control over the quality of the metal made.

Omitting consideration of the details of conversion, manipulation, and utilization of metal made possible by the application of machinery of great power and high efficiency, the simple story of the amount of iron-ore smelted to produce pig-iron between 1870 and 1910 constitutes a sufficient gauge of phenomenal advance; and the fact that in 1871 iron rails commanded $70 and steel rails $102 per ton, while in 1910 no iron rails were produced, and the price of steel rails was $28 per ton, although the prevailing wage-rate had been more than doubled, epitomizes the change of conditions.

Iron-Ore.

The ninth census gave the consumption of iron-ore in the United States for the year 1870 as 3,831,891, and the production of pig-iron as 1,665,179 long tons. At that time, Pennsylvania headed the list of States in pig-iron production, and supplied fully one-third of the iron-ore won. The iron-ore record for 1909 shows: Minnesota, 28,975,149; Michigan, 11,900,384; Alabama, 4,321,252; Wisconsin, 1,067,436; New York, 1,015,333; Virginia, 837,847; and Pennsylvania, 666,889 long tons. The estimated production of iron-ore in the United States in 1910 is 53,500,000, and the pig-iron output was 27,303,567 long tons.

For 17 years prior to 1871, the Marquette range in Michigan had been shipping mineral; but the entire output of Lake Superior iron-ore in 1871 (813,379 long tons) was less than the storage-capacity in 1910 of the 6,918 pockets in the 29 shipping-docks on the great lakes, through which, in that year,

42,619,060 tons of iron-ore were loaded into vessels; and the total production of this range for 17 years was less than its output in 1909. Since 1871, the Marquette range has furnished 93,500,000; the Menominee range (opened in 1877), 75,750,000; the Gogebic range (opened in 1884), 66,000,000, and the State of Minnesota, which up to 1884 had furnished no iron-ore, 256,000,000 long tons. In round numbers, the total production of iron-ore in the Lake Superior region, to the close of 1910, was about half a billion tons—practically all mined during the life of this Institute.

The mining-conditions in 1871 were summarized in a paper by Major T. B. Brooks.[1]

"The iron ores of the Marquette region are mostly extracted in open excavations; hence the process is more properly quarrying. . . . no considerable amount of ore has as yet (1870) been extracted underground in the region, and of that so mined very little has been taken out at a profit; . . . . Nearly the same remarks may be applied to the mines of the Iron Mountain region, Missouri, the ores of which are very similar in character to those of Marquette. Some of the New York and New Jersey magnetic deposits are wrought open, but this is the exception, underground mining there being the rule."

Iron-ores from the Marquette range of Michigan (the only producing section of the Lake Superior region) were then principally used to mix with other ores; and the various sources from which ores were assembled at blast-furnaces, about the time of the organization of the Institute, are suggested by the record that in 1873, eleven blast-furnaces in Pittsburg and vicinity produced 141,773 long tons of pig-iron, and were supplied with ore from the following localities:

	Long Tons.
By rail, Lake Superior ores,	202,840
By rail, Lake Champlain ores,	3,440
By rail, Iron Mountain, Mo., ores,	24,580
By river, Iron Mountain, Mo., ores,	88,489
Native local ores (mostly carbonates),	1,492
Total,	320,841

In 1910, on the other hand, 47 blast-furnaces in the Pittsburg district produced 5,330,982 long tons of pig-iron from 10,000,000 tons of ore brought from the Lake Superior region, practically a ten-fold increase per furnace, and a total district-output augmented 30 times.

[1] *Trans.*, i., 193 (1871–73).

Most of the other Pennsylvania furnaces relied in 1871 on the Cornwall ore-banks or local hematites, while some were dependent on carbonates and Clinton ores. In that year, the Lake Champlain region of New York supplied 183,343 tons of iron-ore, and the New Jersey magnetite-mines about 450,000 tons. The annual output of the New York mines now approximates 1,000,000 tons, and gives promise of material increase, while there has been little change in the total product of the New Jersey mines.

The Ohio furnaces then depended mainly upon local carbonates and Lake Superior ores; but little of the former class is now smelted.

In 1871, the limited amount of ore won in the Southern States fed small charcoal blast-furnaces; but in 1910, Alabama alone made 1,939,147 tons of pig-iron, chiefly from ores developed since the birth of the Institute; and the iron-ore output of Virginia, North Carolina, Georgia, Alabama, and Tennessee now approximates 6,000,000 tons per annum.

Our comparison of the iron-ore and pig-iron industries of 1871 with those of the present day may be emphasized by the mention of some features of special and dramatic interest, such as:

1. The production in one year from a single mine, in Minnesota, of 3,000,000 tons of iron-ore—an amount practically equal to 80 per cent. of the entire output of all domestic mines in 1871.

2. The output of a million tons in 1910 from a single shaft of a Michigan iron-ore mine, raised from the ore-body 2,150 ft. below the surface in skips, carrying 6 tons each, which cover the entire lift in one minute.

3. The practice of digging ore by powerful steam-shovels in large areas, from which 100 ft. or more of over-burden has been stripped; the shipment of this ore in long trains of 50-ton dump-cars; and its transfer into specially-designed vessels through numerous dock-pockets holding 200 to 350 tons each, at a rate which has sometimes exceeded 10,000 tons per hour.

4. The quick voyages of such vessels to receiving-ports and return; the discharge of cargo by mechanical appliances at the rate of 2,000 tons per hour, and the conveyance of the mineral in 50-ton cars to blast-furnaces.

5. The accumulation at docks and at iron-producing plants of stock-piles of ore measured in millions of tons, to be subse-

quently fed at the rate of several thousand tons per day to batteries of blast-furnaces.

6. The increase in magnetic concentration; the mills at one group of mines (Mineville, N. Y.) having a capacity of 3,000 tons per day, while extensive plants have been constructed to treat lean hematites with separators, and nodulizing- and sintering-furnaces.

In the series of operations thus outlined, much of the ore is never touched by the miner, shipper, laborer, or furnace-man, from the time it leaves its natural bed until, with the requisite quantities of flux and fuel, it enters into the charge of the modern blast-furnace, the product of which averages ten times that of the larger furnaces in 1871. Indeed, a considerable portion of the iron-ores now smelted are not touched by the hand of man until, after passing through the blast-furnace, being conveyed by ladle-cars in a molten state to casting-machines, mixers, and converting-plants and mills, they become finished merchantable products.

Pig-Iron.

In 1871 England held first place among pig-iron producing nations, followed by the United States and Germany, but at the present time the output of both the United States and Germany has exceeded that of England; in fact, the United States has surpassed the combined output of Germany and England.

Production of Pig-Iron.

	Production (long tons) 1871	1910.
Great Britain,	6,627,179	10,216,745
Germany and Luxemburg,	1,563,682[a]	14,227,455[a]
United States,	1,706,793	27,303,567

[a] Metric tons.

Notwithstanding the establishment of new iron- and steel-producing centers in other States, Pennsylvania has continued to be the largest contributor of metal. No country in the world (except Germany and Luxemburg combined) made in 1910 as much iron as this State; and the output of the Pittsburg district, notwithstanding the circumstance that it draws the greater part of its ore-supply from sources 800 to 1,100 miles away, was exceeded by no foreign nation except Germany-and-Luxemburg and Great Britain.

The growth of a magnificent industry at cities on or close to the Great Lake system, the establishment of iron- and steel-plants in Alabama and Colorado, and the enlargement of others elsewhere, fall within the interval here contemplated.

The record of the important pig-iron producing States in 1910 was: Pennsylvania, 11,272,323; Ohio, 5,752,112; Illinois, 2,675,646; Alabama, 1,939,147; New York, 1,938,407, and other States, 3,725,932; total, 27,303,567 long tons.

The production of pig-iron by nations and by States could be followed into districts, and the change of status emphasized; for new producing-centers have been added and some old ones have increased in output, while others have been stationary, and a few have shown a decadence. Important factors in these changes have been: (1) the improvements in transportation, which, by increasing the carrying-capacity of vessels and cars, and the efficiency of mechanical handling in loading, unloading, and transfer, have largely eliminated the influence of distance; (2) the concentration of industries under central management; (3) the demand for material in newer sections of the country, creating market-centers, from which the products of furnaces and mills are distributed; and (4) the increased available supply of labor, largely of a skilled character, demanded by the mechanical equipment connected with mines, furnaces, converting-works, and mills.

The marked influences of fuel- and ore-supplies and market-demands upon the establishment of producing-centers have been discussed in other papers which I have presented to the Institute.[2]

The Blast-Furnaces of the 70's.

When the handful of men who, recognizing the advantage of mutual help and interchange of knowledge, assembled in Wilkes-Barre in May, 1871, and organized the Institute, the predominant blast-furnace structure was a truncated square pyramid of stone masonry, lined with refractory brick or stone, the crucible often being formed of stone neatly dressed to shape. From the throats of many furnaces the hot gases, meeting the air, became flame, pulsating with the action of the blast-apparatus and illuminating the surrounding country. Some of the

[2] *Trans.*, xiv., 561 (1885–86); xv., 147, 690 (1886–87); xxi., 473 (1892–93).

newer furnace-stacks, however, were cylindrical shafts of brick held by bands or shells of metal, and supported on masonry piers or metal columns, the top being closed with bell and hopper.

Many furnaces were fed by runways leading to a leveled stock-yard built into an adjacent hill-side; others employed inclined planes; and a comparatively small number used vertical hoists, sometimes water-ballasted, to raise stock from the general working-level of the plant. Iron-pipe hot-blast stoves or long cylindrical boilers (sometimes both) were supported upon costly masonry piers and arches, to facilitate the diversion of the gases from the furnace-top to boilers or stoves.

While some excellent examples of steam blowing-machinery were in use, the prevailing types were horizontal blast-cylinders, operated by spur-gearing from a horizontal steam-engine, or vertical housing or beam-engines of long stroke and large cylinder-diameter, the air-cylinders reaching dimensions of 9 ft. diameter and 9 ft. stroke, and the majority of the blowing-engines being operated without condensers. At some important plants, water-wheels furnished the power; and among the charcoal-furnaces there were examples of wooden blowing-tubs and receivers, the pistons of which were driven by overshot or breast water-wheels.

In the larger furnaces, the general working-limit of blast-pressure was 5 lb. per sq. in.; and if this pressure were doubled, the machinery would be stalled, or the manager would endeavor to loosen up the stock by reducing the burden.

An output of 30 tons per day was considered satisfactory for an average furnace, and the weekly production of 300 tons was sufficient to excite comment. In 1878, the record of 100 tons of pig metal produced by a single blast-furnace in a day, startled metallurgists throughout the world. Closed fronts were a new feature. As a rule, the fluid metal and cinder accumulated in a fore-hearth, the latter overflowing from under a removable plate; and furnaces were "worked" periodically, to remove accumulations of unconsumed fuel, ash, and dirt.

Railway-cars of from 5 to 10 tons capacity delivered the raw material to, or carried the metal from, the more important plants, although some depended largely upon canal-transportation, and many charcoal-furnaces relied solely upon wagon-haul, for raw material and product.

In some large furnaces, masses of coal, ore, and limestone, limited only by the ability of the "fillers" to handle them, were fed into the tunnel-heads, and little attention was given to preparing stock, except at charcoal-plants. The filling of charging-barrows and their discharge into the furnace were done by manual labor; and the casting and breaking of pig-iron demanded a force which practically dominated the operation of the plant; for pig-iron was cast in sand-beds or chills, broken and removed by hand, and cinder was carried away in carts or tram-cars.

In 1870, one-half of the pig-iron product of the United States was made with anthracite coal, 30 per cent. with raw bituminous coal and coke, and 20 per cent. with charcoal; but within five years thereafter, the proportion made with bituminous coal and coke exceeded that obtained with anthracite; and it subsequently increased until, in 1910, the pig-iron output of 27,303,567 long tons was divided into 26,257,978, or 96.2 per cent., made with coke; 649,082, or 2.4 per cent., made with anthracite (generally with coke admixture); and 396,507, or 1.4 per cent., made with charcoal.

To the production of pig-iron should be added that of blooms, averaging about 60,000 tons per year. In 1871, these were made in charcoal-bloomeries from magnetite; but charcoal-blooms are now made from scrap only.

The organization of our Institute occurred at the time when the manufacture of iron was in a state of transition, when the older constructions were being displaced by those of newer design, and the theory of smelting was being scientifically investigated. The situation was epitomized by E. C. Pechin,[3] who said, in a paper, The Position of the American Iron Manufacture, read at the Pittsburg meeting of October, 1872:

"The time has come when scientific research is to assume its true position—the day of 'sheer force and blind stupidity,' whose only protection was a high tariff, has gone by forever. The prodigal waste of the rich gifts of nature; the vast sums of money thrown away; the hard labor, in the aggregate too large to be even approximately estimated, which has been uselessly expended; the mishaps, drawbacks, and failures which have followed every step of our business, show most conclusively that the physicist, the geologist and mineralogist, the chemist, the engineer and mechanic, are as essential to success as the furnace itself, or the labor that works it. . . . . Eternal vigilance is the price of pig-iron."

[3] *Trans.*, i., 279 (1871–73).

Radical Changes in Blast-Furnaces.

In the period under contemplation, there have been radical changes in the shape and proportions, equipment, appliances and location of blast-furnaces. The low flat bosh and narrow crucible were gradually changed, until the "no-bosh" furnace was suggested; and subsequently the very steep slope of boshes gave place to large hearth and moderately flat boshes. The height of furnace, which became excessive, exceeding 100 ft., has settled down to more moderate dimensions. The number and size of tuyères were augmented, and economical blowing-apparatus was designed, to meet the greater demands of volume and pressure. Regenerative hot-blast stoves displaced iron-pipe stoves. The removal of ore- and coke-dust from blast-furnace gases and the cleansing and utilization of these, together with the recovery of the mineral-producing dust, and the employment of gas for operating blowing-machinery and other purposes, as well as the conversion of cinder into cement, and the use of gas from nearly 5,000 by-product ovens, deserve attention in this connection. The production of more than 7,000,000 barrels of cement from blast-furnace cinder, representing about 10 per cent. of the Portland cement output of 1910, and an augmented yield of coke in by-product ovens, accompanied by a recovery of waste products valued at $2,000 per active oven per annum, illustrate the latter proposition.

The various changes in structure, equipment, and operation, the developments in mining, metallurgy, chemistry, and economic management, by which the results mentioned have been obtained, are described and discussed in the cyclopædic library constituted by the 41 volumes of our *Transactions*. In the initiation, investigation, or practical demonstration of these improvements, our members in the United States and other countries have done so much that the progress of the iron and steel industry since 1871 is practically a part of the history of the Institute.

Many to whom the world is indebted for special features of this progress have passed away, leaving as legacies the results of their patient research and ingenuity, while others, who have rendered service of equal value, are still in harness, devoting their energies to economic problems which benefit us all. The

recognition which such men have given to the value of our organization as a medium of the exchange of experiences, indicates the proud position held by the American Institute of Mining Engineers.

While this paper has been confined to the mining and smelting of iron-ores into pig-iron, the efforts of members of the Institute should be recognized in the marvelous improvements made in the conversion of iron into steel, and the fabrication of the metal into merchantable shapes by the use of powerful and economically-operated equipment and machinery, for these have been most potent factors in creating a market for the pig-iron produced. If it were deemed advisable to extend the paper to cover processes beyond the production of pig-iron or to enter into details of mining coal, iron-ore or other mineral, or the treatment of ores other than iron-ores, the services rendered by the members of the American Institute of Mining Engineers would appear as pronounced as in the special lines which have been discussed.

What of the Future?

The wonderful developments of the past 40 years naturally suggest speculation as to the future. It may be that the manufacture of iron and steel is now entering upon an era of radical departure from present practice. The use of electricity for smelting, the advance in magnetic separation and other means of enriching ores, and the nodulizing or sintering of fine material, suggest that some ores now considered undesirable will be in demand, and that deposits now known, but unwrought, will be exploited. Iron-ores now under the ban, because of constituents considered deleterious, may, by beneficiation or improved smelting-methods, be made acceptable. Moreover, the large deposits of iron-ore, notably in the State of New York, and in Cuba and Scandinavia, which require treatment, and those from other countries which reach our ports, indicate a probable revival of the iron industry of our Eastern States, where a liberal market exists. Industrial progress along the Pacific slope, in the Central West, and in the South, also suggests fields for extension of the iron and steel industry, dependent upon raw materials which can be advantageously assembled. The use of dry-air blast; the

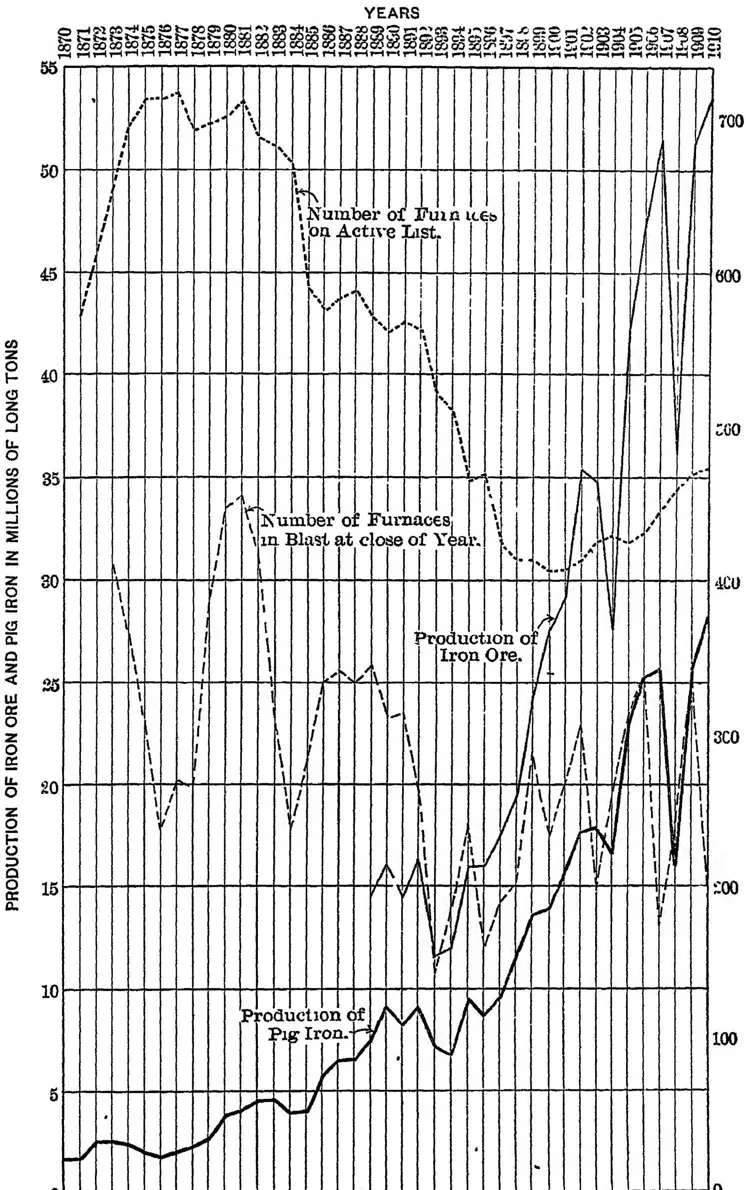

FIG. 1.—RECORD OF THE IRON INDUSTRY OF THE UNITED STATES, 1870 TO 1910.

utilization of blast-furnace cinder as a base for cement-manufacture; the application of gas from blast-furnaces or from by-product coke-plants as a means of power; the recovery of waste; the increase in economy of machinery employed;—
—all these are lines in which further improvement is probable. The extension of labor- and fuel-saving auxiliaries to plants, and the prosecution of chemical and metallurgical research, encourage the hope of a continued production of iron at low cost, while the growing demand for ferro-alloys may develop a radical departure from the present accepted design of plant.

The Graphic Record.

To illustrate graphically the changes in the pig-iron industry of the United States during the last 40 years, the diagram, Fig. 1, has been prepared, in which the ordinates represent years, and the abscissas show on the right the number of blast-furnaces, and on the left the production of domestic iron-ore and pig-iron in millions of tons. The upper curve indicates the number of blast-furnaces reported as active or ready for operation in each year; but it should be remarked that the unwillingness of owners to report a plant as abandoned, makes this number greater than the facts really warrant. While the lower curve shows the number of furnaces in blast at the end of each year, the true condition would in most cases be between the two curves. The decrease in the number of furnaces and the coincident increase in the annual production of pig-iron demonstrate that while the dimensions of the average blast-furnaces of 1910 are much greater than those of 1870, the increased output per furnace far exceeds any increase of size; improvements in equipment, technical management, and scientific metallurgy having raised the average output per furnace from about 5,000 to 100,000 tons per year, to meet a per capita demand of the country augmented six-fold—at the same time greatly reducing the fuel-consumption per ton of product.

The production of domestic iron-ore and pig-iron shows approximately the relations which the raw material bore to the product, but to the quantity of ore should be added mill-cinder, scale, etc., and imported iron-ore, the latter ranging from 180 to 2,591.031 tons per year.

To assist in studying this diagram the figures and quantities are given in Table I.

TABLE I.—*Total Number of Blast-Furnaces in the United States on Dec. 31 of the Following Years, with Domestic Production of Pig-Iron and Iron-Ore.*

Year	Number of Existing Blast-Furnaces	In Blast at Close of Year	Quantity of Pig-Iron Produced.	Quantity of Domestic Iron-Ore Produced
			Tons.	Tons.
				a 3,831,891 (1870)
1871	571		1,706,793	
1872	612		2,548,713	
1873	657	410	2,560,963	
1874	693	365	2,401,262	
1875	713	293	2,023,733	
1876	712	236	1,868,961	
1877	716	270	2,066,594	
1878	692	265	2,301,215	
1879	697	388	2,741,853	
1880	701	446	3,835,191	*a* 7,120,362
1881	716	455	4,144,254	
1882	687	417	4,623,323	
1883	683	307	4,595,510	
1884	669	236	4,097,868	
1885	591	276	4,044,526	
1886	577	331	5,683,329	
1887	583	339	6,417,148	
1888	589	332	6,489,738	
1889	570	344	7,603,642	14,518,041
1890	562	311	9,202,703	16,036,043
1891	569	313	8,279,870	14,591,178
1892	564	253	9,157,000	16,296,666
1893	521	137	7,124,502	11,587,629
1894	511	185	6,657,388	11,879,679
1895	468	242	9,446,308	15,957,614
1896	470	159	8,623,127	16,005,449
1897	423	191	9,652,680	17,518,046
1898	414	202	11,773,934	19,433,716
1899	414	289	13,620,703	24,683,173
1900	406	232	13,789,242	27,553,161
1901	406	266	15,878,354	28,887,479
1902	412	307	17,821,307	35,554,135
1903	425	182	18,009,252	35,019,308
1904	429	261	16,497,033	27,644,330
1905	424	313	22,992,380	42,526,133
1906	429	340	25,307,191	47,749,728
1907	443	167	25,781,361	51,720,619
1908	459	236	15,936,018	35,983,336
1909	469	338	25,795,471	51,294,271
1910	474	206	27,303,567	53,500,000 estimated).

a Census figures.

The number of blast-furnaces and the production of pig-iron are copied from the reports of the American Iron and Steel Association, and the data as to iron-ore are mainly from the statistical reports of the U. S. Geological Survey.

Chamber-Pillars in Deep Anthracite-Mines.

BY DOUGLAS BUNTING, WILKES-BARRE, PA.

(Wilkes-Barre Meeting, June, 1911.)

WITH the gradual exhaustion of the upper veins in the anthracite coal-fields, the problem of mining at greater depths acquires increasing importance and demands the consideration of a number of important factors, one of which is the greater earth-pressure and the consequent necessity of stronger support for the roof.

Under the pillar-and-chamber system, almost exclusively followed in the Northern anthracite-field, the highest economy in mining is generally secured by leaving, on first mining, pillars only sufficient to support safely the overlying strata. As to the necessary size of such pillars, the opinions of mining experts are widely divergent. In establishing the width of chambers and pillars, the thickness of vein and its depth below the surface have received little consideration. For instance, it has been quite usual to work both overlying and underlying veins with the same width of chambers and pillars, when the lower vein was two and one-half times as thick, and twice as far below the surface, as the upper. In view of the generally-accepted theory that the crushing-strength of coal-pillars of the same base-area becomes less with increased height, it is probable, in this instance at least, that the most economical mining has not been secured.

This question of adequate pillar-support is economically less important down to about 800 ft. than at greater depths; and it is with reference to mining at these greater depths that the study of the subject here presented has been prompted.

The necessity of leaving larger pillars, involving greater mining-costs per ton and also smaller yields per acre, is one of the troubles of deep working. To mine without leaving adequate pillar-supports will result, sooner or later, in a squeeze. Limited areas, it is true, have been mined at certain widths of

chambers and pillars, without caving or squeezing; but this is not positive proof that such pillars would be of sufficient size, under the same conditions of thickness of vein and depths below the surface, for larger areas; for frequently a squeeze will not be induced until a large area has been mined. That considerable portions of our coal-deposits have been abandoned, temporarily or permanently, on account of caves and squeezes, is apparent to every observer. The primary cause of a squeeze is insufficient support; the secondary causes are the desire for large immediate output, the lack of systematic mining, and the disturbance of the strata due to some other squeeze or cave. The primary cause, and its possible avoidance, will be considered later. Of the secondary causes, it is unnecessary to discuss at this time the desire for large output, and the lack of systematic mining.

The production of a squeeze by the disturbance of the strata caused by another squeeze, in an overlying or underlying vein, is a very common occurrence, the results of which are frequently as serious as those of the original movement. Workings which, otherwise, would have safely withstood the pressure due to their depth below the surface may be thus disastrously affected. The inherent effects of a squeeze are the crushing of the pillars, the caving of the roof, and the heaving or lowering of the bottom. These occur in various degrees and combinations; but usually the crushing of the pillars is followed by a breaking and caving of the top, which will usually arrest the lateral extension of the squeeze. The area of crushing apparently depends upon the nature and size of the coal-pillars, as well as the nature of the roof. The indirect and general effects of a squeeze include possible loss of life; surface-disturbance, with consequent damage to buildings and other surface-improvements; the liberation of gas and water into the mine; the caving of gang-ways and air-ways; and the necessary suspension of mining in sections of the mine directly affected, and frequently in those contiguous thereto. It need hardly be added that these results, though variable in importance according to local conditions, all add to the costs of mining.

It will doubtless be possible to recover hereafter a considerable part of the coal in old workings where the pillars have

been more or less crushed by the settling of the overlying strata; but this could be done only at increased expense, compared with the present mining-costs, and is therefore not commercially practicable at the present time.

It is, of course, not practicable to determine accurately the unit-pressure on coal-pillars, by reason of the variations in density of the overlying strata. Moreover, the unit-pressure will not vary directly as the depth, according to the law of gravitation; and normal unit-pressure on the pillars, for constant depths and density of overlying strata, will be dependent upon the dip of the vein. However, the variations due to varying densities and the laws of gravitation are so slight for the conditions under consideration that they can be ignored; and the variation of normal pressure due to dip, having little significance for the light dips characteristic of the Northern anthracite-field, may likewise be ignored. We may therefore reasonably say, for the conditions under consideration, that the pressure due to the overlying strata will vary directly as the depth below the surface.

The fracture of anthracite under compression occurs by shearing along planes at various angles to the direction of the applied force. The angle of fracture depends largely upon the brittleness of the coal; and the resistance to movement in these planes is made up of the shearing-strength of the coal, and the frictional resistance along the plane, *i. e.*, the shearing-component of the imposed load. This theoretical angle of rupture has been verified with many materials, but shows in the case of coal considerable variation, probably due to the lack of uniformity of the material. The testing of anthracite coal-specimens for compressive strength presents, therefore, many difficulties, and gives variable figures of crushing-strength. Numerous tests are required for the determination of a fair average for even one size of specimens and one particular vein. The crushing-strength of specimens from the various benches of a vein will vary to a greater or less extent, depending upon the vein. In the preparation of test-specimens, there is difficulty in cutting the specimens to exact dimensions and in obtaining parallel and plane bearing-surfaces. The specimens are liable to contain cracks which are only revealed after loading. All these circumstances influence the results of the test.

Anthracite test-specimens are generally taken from the stronger benches of the vein. The comparative crushing-strength of the various benches of a vein could probably best be arrived at by using drill-cores, which could be cut into desirable lengths.

The compressive-tests on coal-specimens, reported below, were made by Prof. R. C. Carpenter,[1] of Cornell University, and Joseph Daniels,[2] of Lehigh University. Other tests reported in this paper were made by Mr. Daniels on specimens which I submitted.

The relation between the crushing-strength and relative dimensions of Swiss sandstone has been studied very exhaustively by Professor Bauschinger, as stated by Professor Johnson,[3] and, as the result of these tests, he recommends for all shapes of cross-section and relative heights the formula:

$$p = \sqrt{\frac{\sqrt{A}}{\frac{u}{4}}}\left(a + b\,\frac{\sqrt{A}}{h}\right) \quad . \quad . \quad . \quad (1)$$

in which p = crushing-strength per unit area; A = area of cross-section; u = perimeter of cross-section; h = height of specimen; a and b = constants.

A simpler formula for rectangular cross-sections is:

$$p = k + k'\,\frac{b_1}{h} \quad . \quad . \quad . \quad (2)$$

in which b_1 = least lateral dimension; k and k′ = constants.

For the tests on sandstone, referred to above, this formula becomes:

$$p = 5{,}500 + 1{,}565\,\frac{b_1}{h} \quad . \quad . \quad . \quad (3)$$

in which p = crushing-strength in pounds per square inch.

To show the relation between the strength of a prism and that of a cube, Professor Johnson derived, from the results of tests by Bauschinger, the equation:

$$\frac{\text{Strength of prism}}{\text{Strength of cube}} = 0.778 + 0.222\,\frac{b_1}{h} \quad . \quad . \quad (4)$$

in which b_1 = least lateral dimension; h = height of prism.

[1] *Sibley Journal of Engineering*, vol. xvi., No. 3, p. 105 (Dec., 1901).
[2] *Engineering and Mining Journal*, vol. lxxxiv., No. 6, p. 263 (Aug. 10, 1907).
[3] *Materials of Construction* (1897).

The tests on anthracite specimens previously referred to were made on various sizes of cubes and prisms; the cubes varying in size from 2 to 6 in., and the prisms from 2.25 to 12.25 in. in height. These specimens were furnished by the Philadelphia & Reading Coal & Iron Co., the Lehigh Valley Coal Co., Lehigh & Wilkes-Barre Coal Co., Delaware & Hudson Co., and a number of others, and came from numerous veins, with the exception of those of the Lehigh & Wilkes-Barre Coal Co., which were taken from one vein. The results of these tests were tabulated with reference to the size of the specimens and ratio of height to least lateral dimension. The averages were then obtained, and these results are given in Table I.

TABLE I.—*Average Results of Tests on Anthracite Specimens.*

Name of Company	Ratio $\frac{h}{b_1}$.	Crushing-Strength	$\frac{\text{Prism-Strength}}{\text{Cube-Strength}}$.
		Pounds per Square Inch	
P. & R. C. & I. Co.,			
(25 specimens)............	1	2,393	1.00
(25 specimens)............	2	2,296	0.96
L. V. C. Co.,			
(13 specimens)............	1	1,982	1.00
(13 specimens)............	2	1,591	0.80
(13 specimens)............	3	1,405	0.71
L. & W-B. C. Co.,			
(20 specimens)............	0.71	3,025	1.22
(20 specimens)............	1.07	2,566	1.00
(20 specimens)............	1.24	2,393	0.87
(20 specimens)............	1.43	2,008	0.81
(20 specimens)............	1.77	2,090	0.76
(20 specimens)............	2.06	1,880	0.84
D. & H. Co., *et al.*			
(146 specimens)............	0.50	5,113	1.63
(146 specimens)............	1.00	3,131	1.00
(146 specimens)............	2.00	2,234	0.71

These results are plotted in Fig. 1, showing the relation of crushing-strength to $\frac{h}{b_1}$, and in Fig. 2, showing the relation of the strength of a prism to that of a cube, also in reference to $\frac{h}{b_1}$. It is evident, from these plottings, that coal-prisms follow some law of strength relative to their height and breadth.

In Fig. 1 the curve represented by the equation

$$p = 1{,}750 + 750\,\frac{b_1}{h}, \quad . \quad . \quad . \quad (5)$$

which is based on a crushing-strength of 2,500 lb. per sq. in. for cubes, is plotted to show how well it fits the average results of the tests. In this figure there are also indicated, by concentric circles, the calculated pressures per square inch on chamber-pillars which have caused squeezes, and with which I am more or less familiar. In consideration of these unit-pressures, a factor of safety is arrived at for practice; and the curve in Fig. 1 represented by the equation

$$p = 700 + 300 \frac{b_1}{h} \quad . \quad . \quad . \quad (6)$$

is taken for that purpose. This gives a factor of safety of 2.5 for cubes with a crushing-strength of 2,500 lb. per sq. in., which is a fair average of the tests.

In Fig. 2 the curve represented by the equation

$$\frac{\text{Strength of Prism}}{\text{Strength of Cube}} = 0.70 + 0.30 \frac{b_1}{h} \quad . \quad . \quad . \quad (7)$$

is taken as best representing the results of these tests and other considerations. In Fig. 2 the equation

$$\frac{\text{Strength of Prism}}{\text{Strength of Cube}} = 0.778 + 0.222 \frac{b_1}{h}, \quad . \quad . \quad . \quad (8)$$

showing the law of variation of relative strength of prisms and cubes of sandstone, as evolved by Professor Johnson, is also plotted.

To arrive at a formula for proportioning pillars in the pillar-and-chamber system of mining, the weight of the overlying strata is taken at 144 lb. per cu. ft., with the following notation, all in feet:

y = depth below the surface; b_1 = width of pillar; z = distance from center to center of chambers; h = total thickness of vein.

The load per square foot on a pillar will be: $\frac{144\, y\, z\, b_1}{b_1^2}$,

And with 1,000 lb. per sq. in., or 144,000 lb. per sq. ft., as the safe loading for a cube we obtain by substituting in equation (7)

$$\frac{144 y z b_1}{b_1^2} = 144{,}000 \left(0.70 + 0.30 \frac{b_1}{h}\right)$$

$$\text{or} \qquad yz = 1{,}000 \left(0.70 + 0.30 \frac{b_1}{h}\right) b_1 \quad . \quad . \quad (9)$$

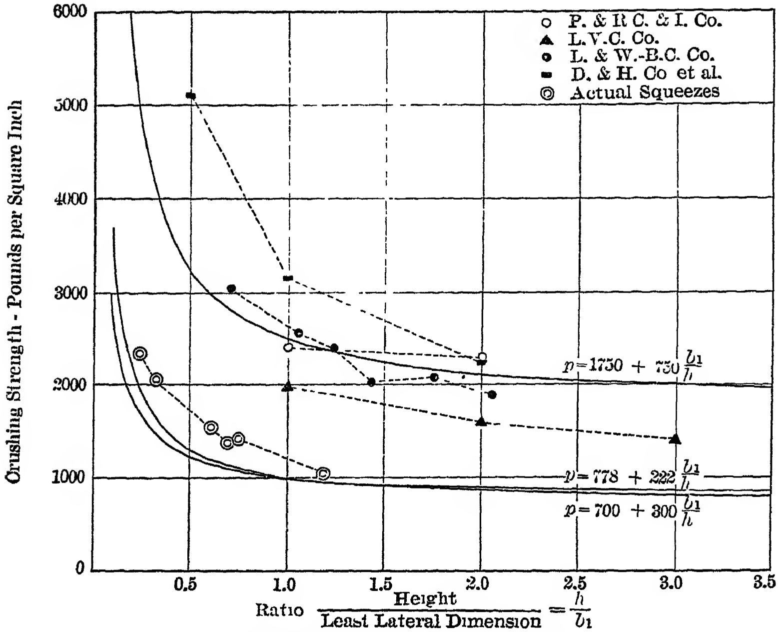

FIG. 1.—RELATION BETWEEN THE CRUSHING-STRENGTH AND THE RATIO OF HEIGHT TO LEAST LATERAL DIMENSION.

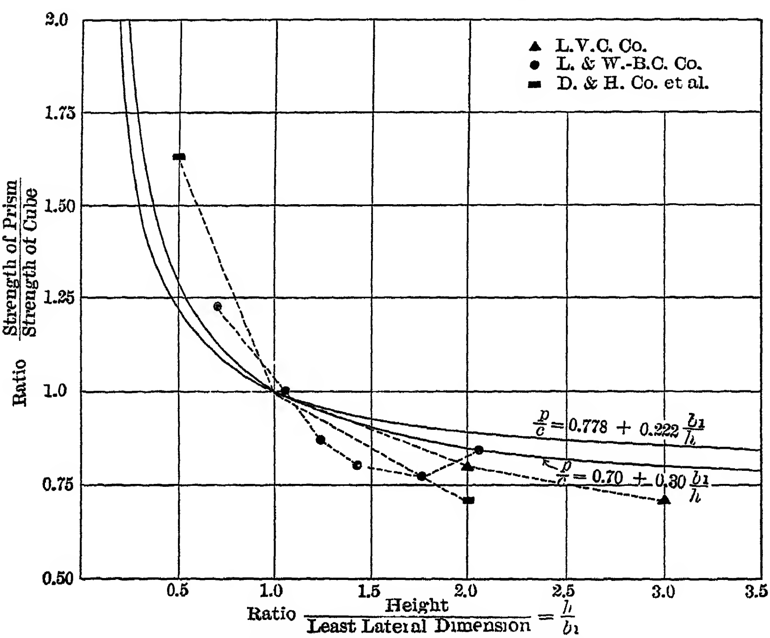

FIG. 2.—RELATION BETWEEN THE CRUSHING-STRENGTH OF PRISMS AND CUBES.

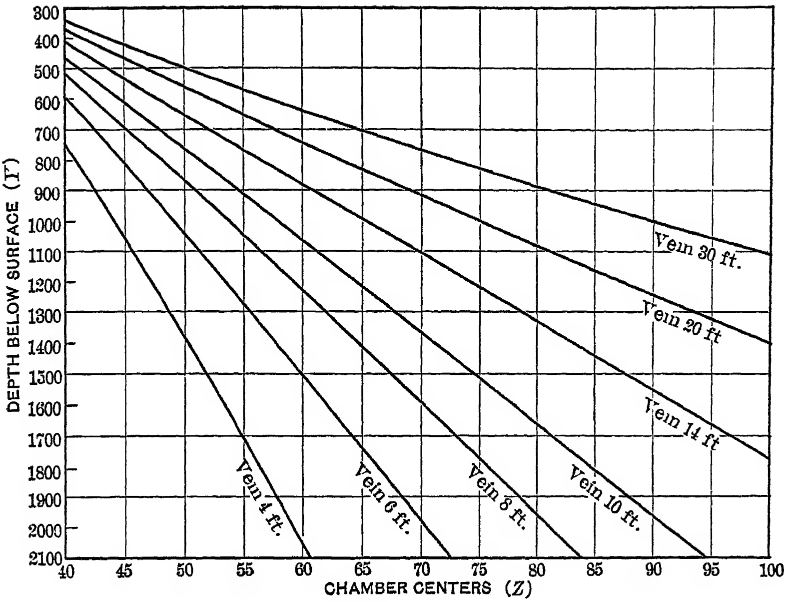

FIG. 3.—THE RELATION BETWEEN DEPTHS AND CHAMBER-CENTERS FOR VARIOUS THICKNESSES OF VEINS FOR 24-FT. CHAMBERS.

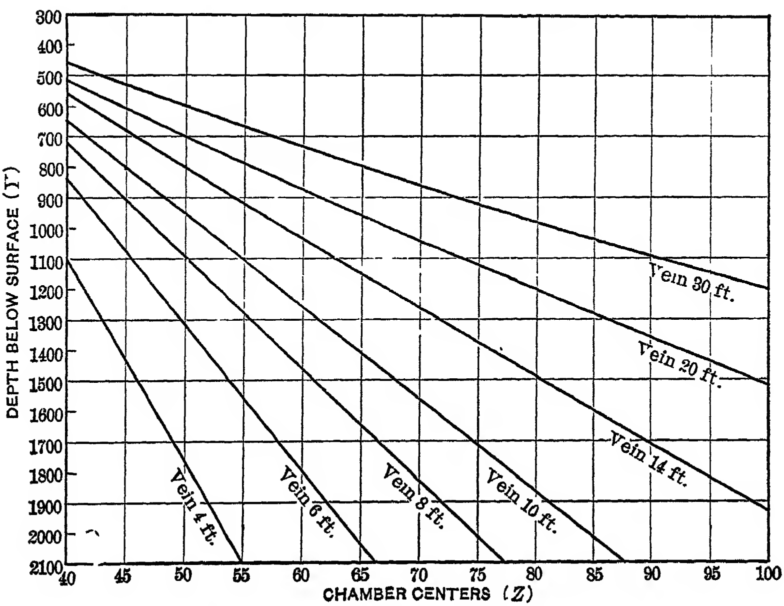

FIG. 4.—THE RELATION BETWEEN DEPTHS AND CHAMBER-CENTERS FOR VARIOUS THICKNESSES OF VEINS FOR 20-FT. CENTERS.

Should the "checker-board" system of mining, which is a modification of the pillar-and-chamber system, be used, the equation would become:

$$yz = 1{,}000 \left(0.70 + 0.30 \frac{b_1}{h}\right) b_1^2 \quad . \quad . \quad . \quad (10)$$

Tables II. and III., diagrammatically represented in Figs. 3 and 4, show the relation between the depths and chamber-centers for various thicknesses of veins, and were prepared from equation (9); Table II. giving these relations for 24 ft., and Table III. for 20 ft. width of chambers. I prepared Table IV. several years ago from the equation:

$$yz = 1{,}000 \left(0.778 + 0.222 \frac{b_1}{h}\right) b_1 \quad . \quad . \quad (11)$$

which is based on the crushing-strength of coal-specimens at the first indication of failure, and a safe crushing-resistance, for cubes, of 1,000 lb. per sq. in. It is evident that, for the lower ratios of height to width of pillars, this equation gives higher factors of safety than equation (9). For practical use, however, I believe that equation (9), from which Tables II. and III. were derived, is better for average conditions of veins and light dips.

The question of dip is important; and it is to be understood that the suggestions here given are applicable only to the light dips characteristic of the Northern anthracite-field; but I hope to give further consideration at some future time to this subject in its relation to heavy dips.

In the application of any formula to the calculation of the size of pillars necessary to resist safely the pressure of the overlying strata, consideration will have to be given to a number of conditions, such as the nature of the vein, as well as its contiguous stratum, and the dip. Moreover, the factor of safety will be dependent upon local conditions, such as the relative location and extent of workings, and the seriousness of possible disturbance to the overlying strata and surface. In conclusion, I wish to say that I have derived a formula for shaft-pillars, based on the same line of deduction, which will be presented at some future time. Meanwhile, I trust that the foregoing suggestions will incite other members of the Institute to offer their opinions on this subject, which will be of much value to all concerned.

TABLE II.—*Depths Below Surface for Various Chamber-Centers and Thicknesses of Veins, 24-ft. Centers.*

		Thickness of Vein													
z	b_1	4	6	8	10	12	14	16	18	20	22	24	26	28	30
40	16	700	600	520	472	440	417	400	387	370	367	360	354	349	344
45	21	1,062	817	694	621	572	537	510	490	474	460	419	440	432	425
50	26	1,378	1,040	871	770	702	654	617	589	567	548	533	520	509	499
55	31	1,705	1,268	1,050	919	831	768	722	686	657	633	613	596	582	569
60	36	2,040	1,500	1,230	1,068	960	884	825	780	744	714	690	669	651	636
65	41	2,381	1,735	1,411	1,217	1,088	996	926	873	830	794	765	740	719	700
70	46	2,727	1,971	1,596	1,367	1,216	1,108	1,027	964	913	872	838	809	784	762
75	51	3,077	2,210	1,777	1,516	1,343	1,219	1,126	1,054	996	949	909	876	817	823
80	56	3,430	2,450	1,960	1,666	1,470	1,330	1,225	1,143	1,078	1,025	980	942	910	882
85	61	3,786	2,691	2,144	1,816	1,597	1,440	1,323	1,232	1,159	1,099	1,049	1,007	971	940
90	66	4,143	2,933	2,328	1,965	1,723	1,550	1,421	1,320	1,240	1,173	1,118	1,072	1,032	997
95	71	4,503	3,176	2,513	2,115	1,849	1,660	1,518	1,407	1,319	1,247	1,186	1,136	1,092	1,054
100	76	4,864	3,420	2,698	2,265	1,976	1,770	1,615	1,495	1,398	1,320	1,254	1,198	1,151	1,110

TABLE III.—*Depths Below Surface for Various Chamber-Centers and Thicknesses of Veins, 20-ft. Centers.*

		Thickness of Vein													
z	b_1	4	6	8	10	12	14	16	18	20	22	24	26	28	30
40	20	1,100	850	725	650	600	565	538	517	500	486	475	465	457	450
45	25	1,431	1,083	910	806	736	687	649	620	597	578	562	549	538	528
50	30	1,770	1,320	1,095	960	870	806	757	720	690	665	645	627	613	600
55	35	2,116	1,559	1,281	1,114	1,002	922	863	817	779	748	724	703	684	668
60	40	2,467	1,800	1,467	1,267	1,133	1,038	967	912	867	830	800	774	753	733
65	45	2,821	2,042	1,652	1,419	1,264	1,152	1,069	1,003	952	909	874	844	818	796
70	50	3,178	2,285	1,839	1,571	1,393	1,265	1,169	1,095	1,036	987	946	912	883	857
75	55	3,538	2,529	2,025	1,723	1,522	1,377	1,270	1,186	1,118	1,063	1,017	978	945	917
80	60	3,900	2,775	2,212	1,875	1,650	1,489	1,369	1,275	1,200	1,139	1,088	1,044	1,007	975
85	65	4,263	3,021	2,399	2,027	1,778	1,600	1,467	1,364	1,281	1,213	1,156	1,109	1,068	1,033
90	70	4,628	3,267	2,586	2,177	1,906	1,711	1,566	1,452	1,361	1,287	1,225	1,174	1,128	1,089
95	75	4,994	3,513	2,773	2,329	2,096	2,033	1,663	1,539	1,441	1,360	1,293	1,236	1,187	1,145
100	80	5,360	3,760	2,960	2,480	2,160	1,931	1,760	1,627	1,520	1,433	1,360	1,298	1,246	1,200

TABLE IV.—*Depths Below Surface for Various Chamber-Centers and Thicknesses of Veins, Calculated from Equation* (11).

		Thickness of Vein.													
z	b_1.	4	6	8	10	12	14	16	18	20	22	24	26	28	30
40	16	666	548	489	453	427	413	400	390	382	376	370	366	362	358
45	21	907	726	635	581	539	518	499	484	472	462	454	447	441	436
50	26	1,155	905	779	705	655	618	592	571	554	541	530	520	512	504
55	31	1,408	1,085	923	826	762	716	681	654	632	615	600	587	577	568
60	36	1,666	1,266	1,066	946	866	811	766	733	706	684	666	651	638	626
65	41	1,926	1,448	1,209	1,065	969	901	850	810	777	751	730	711	695	682
70	46	2,189	1,630	1,350	1,182	1,070	992	931	884	846	816	790	769	751	735
75	51	2,454	1,812	1,492	1,299	1,170	1,079	1,010	957	914	880	850	825	804	780
80	56	2,720	1,995	1,632	1,415	1,270	1,166	1,088	1,028	980	938	906	880	855	833
85	61	2,988	2,178	1,773	1,530	1,369	1,252	1,161	1,098	1,044	1,000	963	932	905	880
90	66	3,257	2,361	1,913	1,645	1,466	1,338	1,242	1,167	1,107	1,060	1,017	983	954	928
95	71	3,525	2,545	2,054	1,759	1,563	1,428	1,318	1,236	1,187	1,132	1,086	1,047	1,014	985
100	76	3,797	2,728	2,196	1,873	1,660	1,507	1,392	1,303	1,232	1,173	1,125	1,084	1,049	1,019

Mine-Caves Under the City of Scranton.

BY ELI T. CONNER, PHILADELPHIA, PA.

(Wilkes-Barre Meeting, June, 1911.)

MY connection, under a commission from the Councils and Board of School Control of the city of Scranton, Pa., with a recent investigation of mine-caves and the resultant damages to surface-improvements, has led to the preparation, at the invitation of our Secretary, of the present paper.

It is notorious that there are, in the anthracite-fields, frequent subsidences of the surface, due to the removal of the coal beneath. No particular attention is paid to such occurrences, unless they happen to injure surface-improvements. Many caves have happened in Scranton and its vicinity, which have excited but little remark, since they have generally caused no serious damage.

In August, 1909, a cave occurred in the Hyde Park section of the city, generally known as the West Side, which nearly destroyed school-house No. 16 and considerable adjacent property. Fortunately, there were no pupils in the building at the time; but the thought of the possible result, had the usual number of pupils and teachers been present, aroused the public to the gravity of the situation; and the School Board employed engineers to investigate the case. Two reports were made by separate sets of engineers, which differed in some particulars. These differences were pointed out in the public press and magnified, and the consequent agitation of the subject was taken up and greatly exaggerated by some of the metropolitan newspapers, giving to uninformed people the very erroneous impression that the whole city of Scranton was in danger of sinking into the bowels of the earth.

All this tended to affect the credit of the city and to depress real-estate values. The matter was considered at a joint meeting of the Board of Trade, the Councils, and the School Board. The Hon. J. Ben. Dimmick, a former Mayor, suggested that it

would be desirable to ascertain the true state of mining-conditions under the whole city, and proposed that an advisory board of disinterested engineers of national repute be invited to assist the authorities in such an inquiry. Accordingly, Messrs. John Hays Hammond, W. A. Lathrop, D. W. Brunton, L. B. Stillwell, and R. A. F. Penrose were thus invited, and recommended that Messrs. William Griffith, of Scranton, and Eli T. Conner, of Philadelphia, be employed by the city and School Board to inspect the mines under the city and report on the actual conditions, after submitting their findings to the Advisory Board. This plan was adopted by the city and school authorities, and the examining engineers began work in October, 1910. During the investigation we took a number of photographs of the conditions observed, some of which are here presented.

In order to illustrate the present extent of the mine-workings under Scranton, maps were made, using as a basis the City Atlas, containing 24 plates. These plates were traced, showing all the streets, alleys, etc., as also all of the school-houses, churches, public buildings, street railway-lines, streams, and railroads. Fig. 1 shows the plate embracing the central part of the city.

The method adopted for showing the worked-over area in the several beds of coal, is by dotted lines and dashed lines at varying angles, as shown by nomenclature on the bottom border of the map. The cross-section on the upper border of the map shows the workable beds of coal.

The pillars shown on the cross-section are only conventionally represented. In our examination of the mines and maps, it was found that in nearly all beds, irrespective of thickness or depth from the surface, the old empirical rule of leaving about one-third of the coal as pillars had been followed, except under the South Side, where considerably less has been left; probably not over 20 per cent. The cover over the one or two seams mined in this section, however, is not great, which accounts for its not having caved hitherto.

The uppermost bed shown, known as the Fourteen-Foot or Big Vein, was mined many years ago, and the workings are largely inaccessible at present.

The next bed, known as the Clark, is worked over a larger

area of the city than any other, and in the section of the city shown on this plate the pillars are small, and in many places seriously "chipped."

The lower four beds shown on this plate, known as Dunmore Nos. 1, 2, 3, and 4, developed under a large part of the city, are now being mined. These beds, being thin, require the removal of top or bottom for height. This rock, together with the waste material in the bed proper, is usually stowed on one or both sides of the road, affording some reinforcement to the pillars; but since it is deposited very loosely, it cannot be considered as an effective support.

West of the Lackawanna river the conditions are quite different from those found under the East Side, as there are more workable beds of coal, several of them being quite thick and close together.

Figs. 2 and 3 show two plates of the Atlas covering portions of the city west of the river. Attention is directed to the Diamond, Rock, Big, and New County beds, Fig. 2, which here aggregate about 45 ft. within about 160 ft. between the roof of the uppermost and the floor of the lowest bed of the series. These were the seams first attacked in the early days of mining, and no care was exercised to columnize the pillars; *i.e.*, to locate them over each other in the several beds mined. As a consequence, the weight of the over-burden, occasioning great complexity of strains on the intervening strata, has been, and will hereafter be, a fruitful source of caving. It was this condition that, in our opinion, caused the cave which affected School No. 16. Attention is directed to No. 25 school, to which reference will be made later. This is shown by the numerals "25" on Fig. 2.

Fig. 4 shows "chipping pillars" in the Clark seam under the central part of the city. This is unmistakably due to the fact that the pillars are too small to sustain the over-burden, and is the usual first sign of what will eventually be a complete collapse of the coal-pillars, and fall of the roof. If the pillars are left undisturbed, the pressure upon them may continue for a long period before appreciably affecting the overlying strata. Pillars sometimes "chip" from exposure to the air, which is known as "air-slack;" but we do not think this to have been the cause of the chipping shown on this picture,

which is, on the contrary, undoubtedly due to excessive weight.

In every coal-seam there is one bench softer than any other part of the seam. In the Clark seam, the bench near the mid-

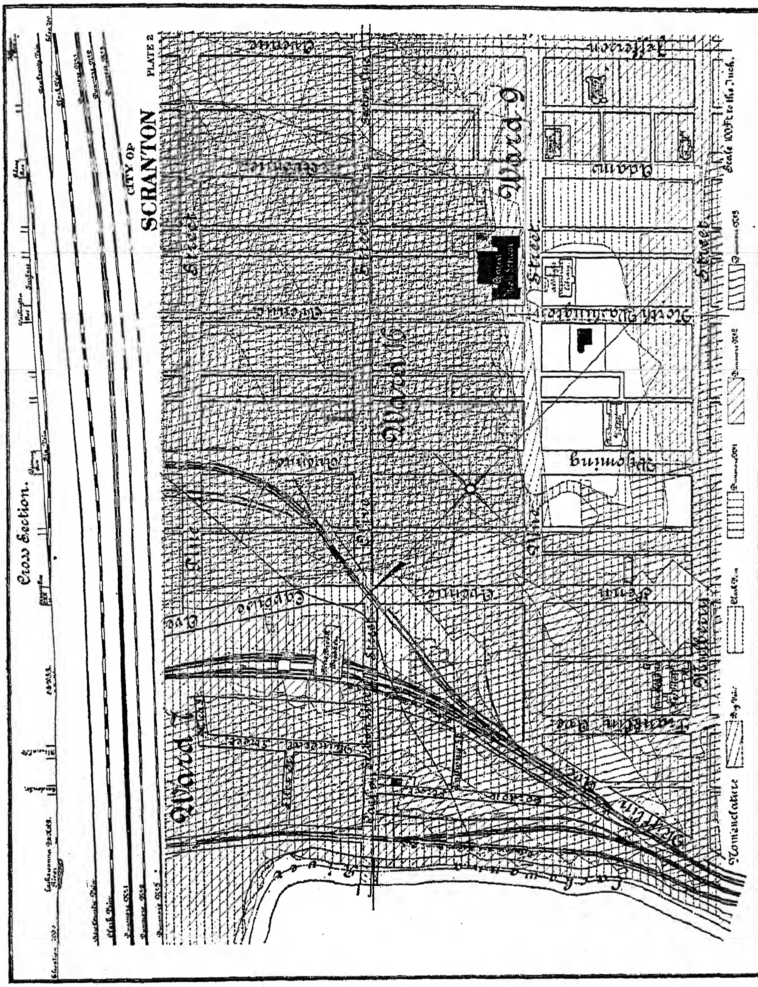

Fig. 1.—Showing Worked-Over Area in the Several Coal-Seams.

dle is the weak part, where the map and note-book are stuck behind flakes of coal, as shown in the picture. The Clark seam here is between 6 and 9 ft. thick.

While engaged on this investigation a large school building,

No. 25, in the Providence section of the city, showed evidence of settling, and it was deemed necessary to close it. We were asked by the School Board to make a special investigation, and advise them as to the cause.

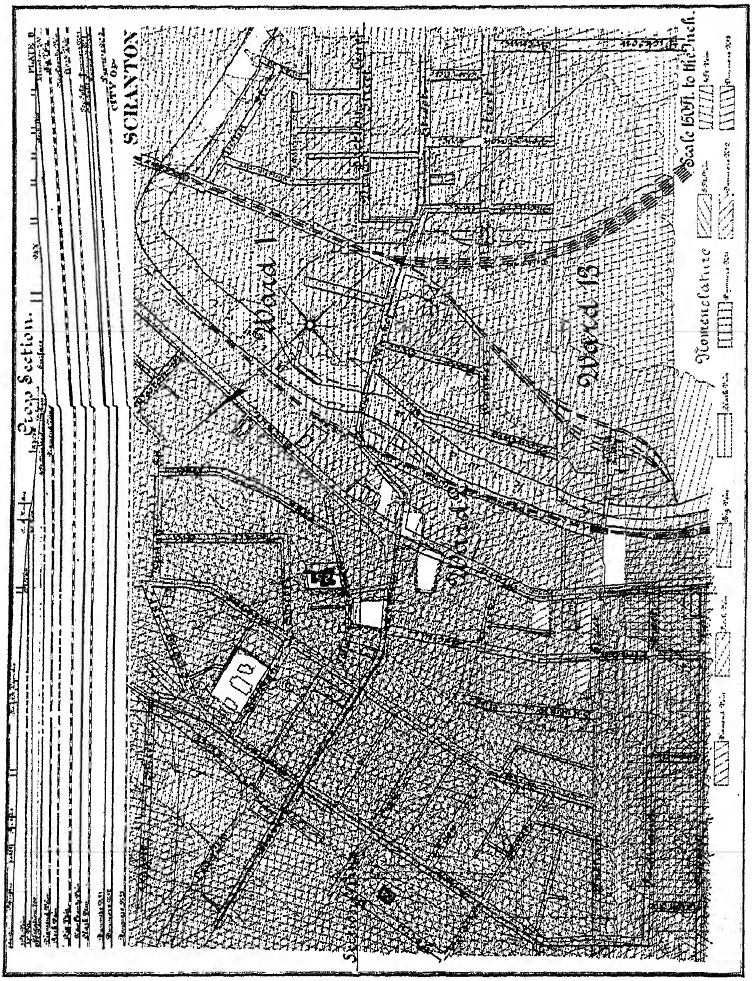

FIG. 2.—SHOWING WORKED-OVER AREA IN THE SEVERAL COAL-SEAMS.

It was found that Dunmore bed No. 4, about 700 ft. below the surface, was "creeping" or "squeezing." This seam is about 4.5 ft. thick. The pillars left were not over 33 per cent. of the original bed. No signs were observed of what is commonly

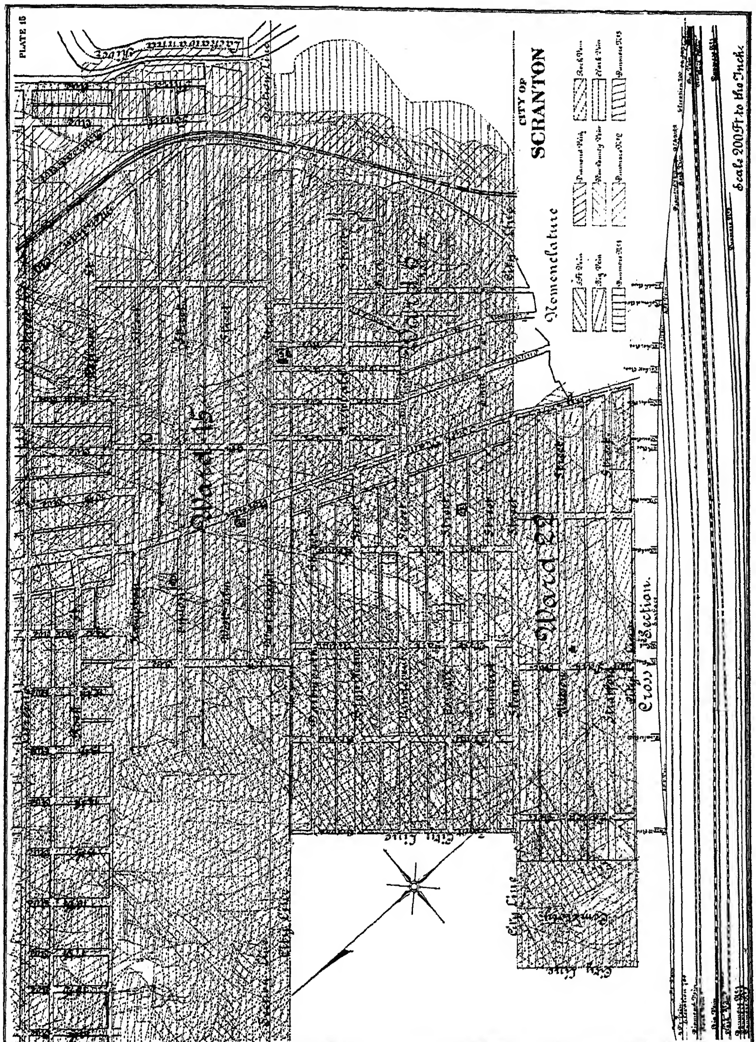

FIG. 3.—SHOWING WORKED-OVER AREA IN THE SEVERAL COAL-SEAMS.

FIG. 4.—CHIPPING COAL-PILLAR.

FIG. 5.—EFFECTS OF SQUEEZE UNDER NO. 25 SCHOOL.

FIG. 6.—RE-MINING BOTTOM COAL, DIAMOND BED.

FIG. 7.—RE-MINING BOTTOM COAL, BIG OR FOURTEEN-FOOT BED.

FIG. 8.—GOB-PIER UNDER CENTRAL HIGH SCHOOL.

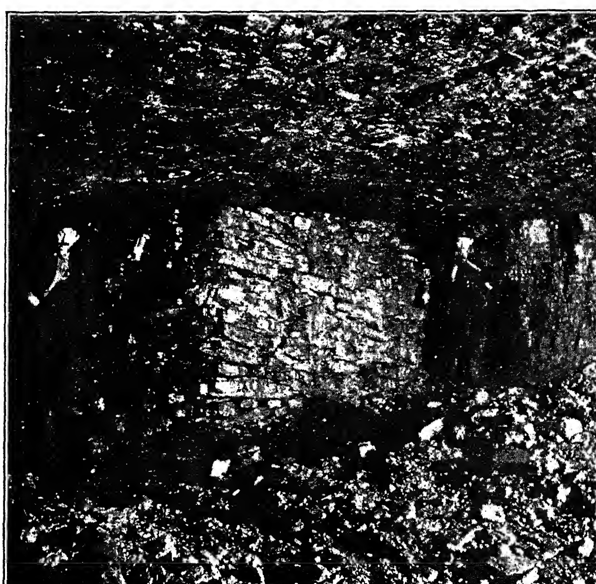

FIG. 9.—DRY GOB-PIER UNDER NO. 10 SCHOOL.

FIG. 10.—SANDSTONE AND CEMENT PIER UNDER NO. 15 SCHOOL.

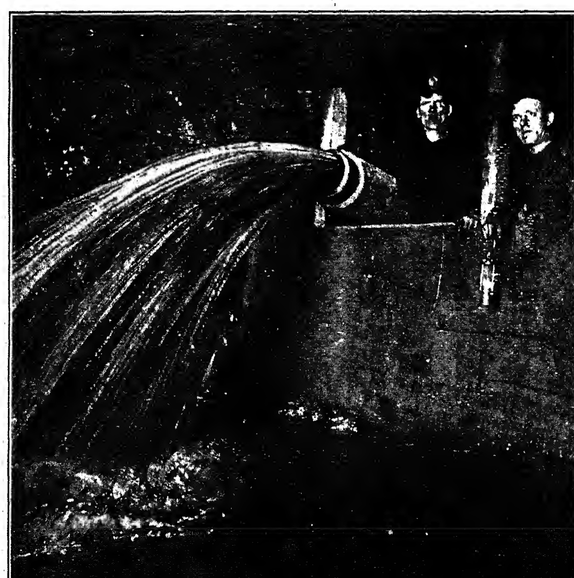

FIG. 11.—FLUSHING-PIPE DISCHARGE.

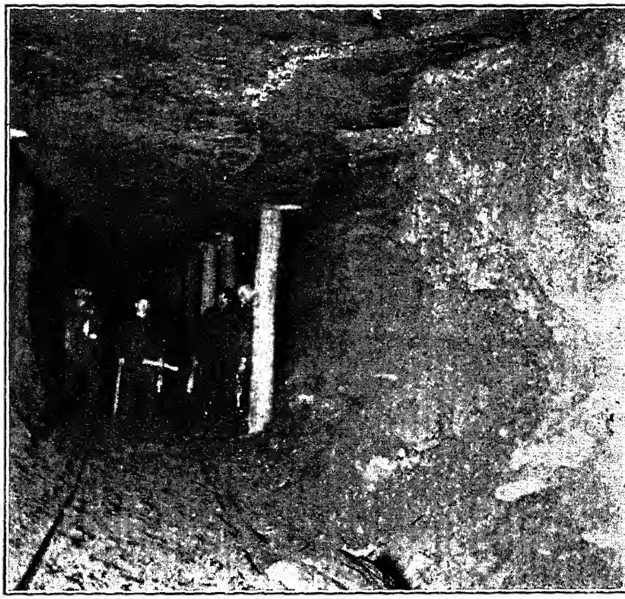

FIG. 12.—REOPENED CULM-FLUSHED GANGWAY.

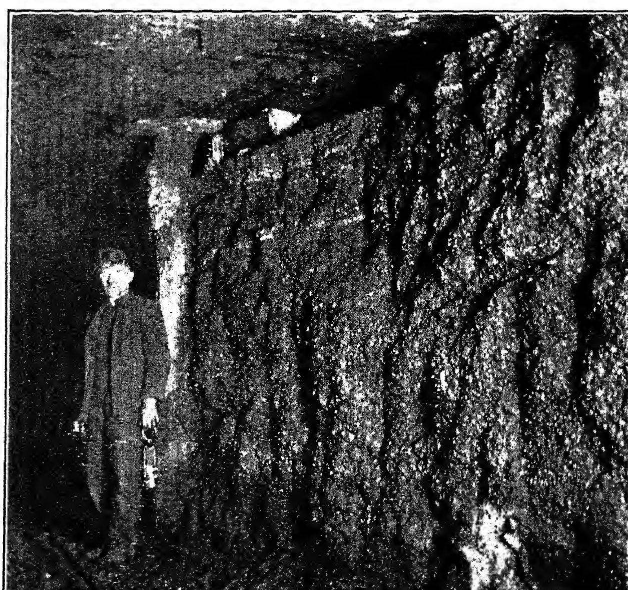

FIG. 13.—REOPENED CULM-FLUSHED GANGWAY; CULM NOT ROOFED.

called robbing of pillars, but owing to their smallness and the great depth, a squeeze had started, and there being no large blocks of coal to stop it, there is every likelihood of its continuing until roof and floor practically come together, as we observed in the same mine at another point.

Fig. 5 shows the effects of the squeeze that disturbed School No. 25. This picture was taken under the school. This opening has since been closed, and is now inaccessible.

School No. 44 is located over an area that is now closed in the Dunmore No. 4 bed by a cave similar to the one that damaged School No. 25; but the whole of the area under No. 44 was mined in the usual manner; so that all of the overburden settled gradually and uniformly, while School No. 25 is half-supported on a solid pillar in all the beds beneath it. Consequently, one-half settled and the other could not, resulting in the damage to the building. From observation of this and other instances, we formed the opinion that the reservation of small blocks of coal in deep-lying and comparatively thin beds is detrimental, rather than beneficial, to surface-improvements.

In the first mining of the Diamond and Big beds, the bottom benches of coal, from 3 to 6 ft. thick, with partings of slate and bone, were not taken. In recent years the old workings have been reopened to recover this coal. Figs. 6 and 7 show this re-mining. The removal of this bottom-coal makes the openings 16 to 24 ft. high, consequently weakening the original pillars.

Note picture of Big bed (Fig. 7). The portion of seam from the fire boss's feet upward was original mining, and that below his feet is recent re-mining. In this vicinity the New County bed, from 5 to 8 ft. thick, is but 6 ft. below the floor of the Big bed, as shown by the cross-section in Fig. 3.

Fig. 8 shows a gob-pier built under the Central High School. This pier is pointed with good mortar, and appeared quite strong. But a hole which was broken through the wall for the purpose of carrying a culm-pipe to points beyond, revealed that the inside of the wall or pier is simply loose rock laid up dry; so that as a support this pier is very limited in bearing-strength. Fig. 9 shows a dry gob-pier built of slate, bone, and fire-clay, under School No. 10. Many such piers have been constructed under valuable surface-improvements. Their low efficiency as

an effective support is apparent. We, however, hesitated about expressing this opinion officially, without having some definite data as to the compressibility of such material so laid up. As there was no information available on this point, we determined to make a series of tests at the superb Fritz Engineering Laboratory at Lehigh University, to which reference will be made below. Fig. 10 shows a pier under School No. 15, which is undermined in the Dunmore No. 2 seam, here only 25 ft. from the surface. Sixteen of these piers were constructed of sandstone brought in from the surface, and laid up in good cement-mortar, making an effective support.

Fig. 11 shows the method of "flushing" with "culm," *i.e.*, small coal, ground slate, and other refuse, washed through pipes into the underground openings. The chambers are closed with a plank battery at the lower end, and as they fill, the water seeps away and flows to the sump. Fig. 12 shows a gangway reopened through a culm-filled area. It will be noticed that the sides are smoothly vertical, showing that the culm packs thoroughly, and, when roofed, makes an effective support. Considerable areas under the city of Scranton, and at most anthracite collieries elsewhere, are thus filled, this being now a generally-accepted method of disposal of refuse.

Fig. 13 shows the same culm-filled and reopened gangway further along, where the culm is not roofed. In this figure, the fire-boss is seen on the top of the culm, which at this point is about 16 in. from the roof. The supporting-value of this culm is, of course, decreased by its not having been properly roofed.

The results of all tests made are given in Tables I. to IV.

Table II. exhibits the value of the various devices for dry filling, and also the value of the different materials available for flushing at coal-mines in this locality. The figures are directly deduced from the results of the tests made at Lehigh University, and we think are sufficiently clear to be self-explanatory. We might add, however, that test No. 1 represents the value of well-constructed gob-piers, and Nos. 6, 7, and 8 show the supporting-value of mine-rooms filled with rock blasted from the floor and roof, as heretofore mentioned; while Nos. 12 and 13 indicate the supporting-strength of fine material, such as coal, culm, and river-sand, flushed in with water. At the bot-

tom a comparison is made as between the supporting-value of the flushed culm and the flushed sands and the concrete piers of the same nature, as per samples tested.

The approximate cost, per foot of bed-thickness, for each acre of complete flushing under schools and elsewhere, and to take the place of pillars, if removed, would be

For culm, below level of river, $405.00
For sand, above or below river, $1,615.00

A factor of safety of 2 was used in arriving at the above costs. This, I think, is excessive; and I believe that the work could be done at smaller expense.

These tests were made to determine the efficacy of the various methods of roof-support heretofore used, as well as to discover a comparatively inexpensive combination of materials which might be more efficient and permanent, and might possibly permit of the recovery of the major portion of the remaining pillar-coal. But it is not claimed that the calculated strength of artificial supports shown by the tests for the subsidences indicated should be taken as final. These results ought to be checked by actual experience wherever possible.

I have inspected one mine, where a series of about 18 chambers, 30 ft. wide by about 400 ft. long in a seam 5.5 ft. thick, had been flushed with culm properly roofed. Later, the remaining pillars, about 18 ft. thick, were removed one at a time, and the spaces were immediately flushed with culm. The roof in this case did not seriously crack, but simply settled bodily upon the culm, showing an average subsidence of about 7 in., or approximately 10 per cent. As this seam of coal was about 500 ft. below the surface, this experience corresponds fairly with the calculated subsidences shown in the foregoing tables, as well as with my experience in charge of similar pillar-recovery.

The conclusions drawn from our inspection of mining and geological conditions, and our tests of materials, were: (1) that flushing with culm, crushed rock, or sand is practically the only proper and available method for the support of overburden, and the ultimate recovery of the pillar-coal; and (2) that under any circumstances, some subsidence of the surface must be expected, depending on the thickness of the seams of coal completely extracted, and their depth below the surface.

With regard to possible sources of the supply of sand, which is, from every point of view, by far the best flushing-material, it is my belief that the large body of sand known to overlie the coal-measures in the Wyoming valley from Pittston to Nanticoke, would not be suitable, since it is mostly quicksand, and, if flushed into the mines, could probably not be confined and drained, as is necessary for successful operations of this kind. But I believe that large quantities of sand could be brought from distant points in returning empty railroad-cars at comparatively slight cost. Another source of supply is the establishment of efficient crushing-plants of large capacity, on the hill-sides near the outcrops of the several beds of coal, to pulverize the rock that is available for this purpose.

Finally, I regard the conclusions deduced from the tests made, and the calculations and tabulations based thereon, as reasonably reliable; yet I would record the opinion that some of the collieries present conditions to which they might not apply—for instance, in localities where several seams of coal are separated by thin strata of shale and slate, or even sandstone, and the pillars in the two or more seams are not over one another, and it is proposed to reclaim all or any part of the pillars. In such a case, even though the following tables might seem to be applicable, I think the only permissible procedure would be, first to fill with flushed material all the openings in the lowest seam of the series, and then to continue the process upward until all are filled, care being taken to have the flushed areas over one another. After all the openings in all the seams have been filled, the pillars in the uppermost seam may be attacked; and, as each pillar is removed, the space thus left should be filled before the next pillar is removed. No pillar-reclamation should be permitted in any of the other beds until all of the pillars in the upper bed have been removed, and the over-burden has come to rest on the flushed material. After this, the pillars in the next lower seam may be attacked and handled in like manner.

TABLE I.—*Compression-Test on Various Materials Used for Mine-Support.*

No. of Test.	Description of Test.	Net Tons Per Square Foot Required to Produce Compression of 1 Per Cent.	3 Per Cent.	5 Per Cent.	10 Per Cent.	20 Per Cent.	30 Per Cent.	At End of Test	Remarks.
1.	Rectangular piers of mine-rock,	.....	0.8	1.4	2.7	9.5	23.0		Average construction. Voids not filled
2.	Circular pier of mine-rock,		3.5	5.67	11.0	22.0	38.5	31 per cent settlement 42.5	Well constructed. Voids filled with small and shovel material
3.	Timber crib filled with mine-rock,		0.6	1.37	5.11	20.3	31.5		Average construction
4.	Pile of broken sandstone; small sizes,		.	.	4.0	9.3	22.4	46 per cent settlement 104.5	In these tests the material was not confined, but was free to expand laterally
5.	Pile of small size broken sandstone and sand,	.	...	1.6	4	13.5	35	35 per cent. settlement 144.0	
6.	Pile of broken sandstone, large sizes,	..	3.6	5.0	9.1	26.4	37	41 per cent settlement. 75.0	
7.	Pile hard coal-measure sandstone similar to No. 6,	0.09	2.1	3.5	9	33.4	27 per cent. settlement 55		
8.	Pile of river-sand,							63 per cent. settlement 108.0	
9.	Broken sandstone in cylinder,		3.33	5.55	13.22	46.6	98.6	35 per cent settlement 666.0	
10.	Broken sandstone and sand in cylinder,		3.5	5.77	24.42	308.5	23 per cent. settlement. 666.0		In these tests the material was confined and could not expand laterally
11	Dry coal-ashes in cylinder, Coal-ashes flushed in with water,		1.0 .	1.86 .	5.32 .	10.8 5.50	25.0 22	51 per cent settlement. 666.0	
12.	Wet culm flushed in cylinder; partly dried,	2.44	Average of two tests. 8.9	14.28	35.52	138.7	444	33 per cent settlement 666.0	
13.	Dry sand in cylinder,	0.88	3	5.27	33.3	129	499	32 per cent 666	
14.	Wet sand flushed and partly dried	8.4	39.3	67.0	173.8	555.4	20.75 per cent settlement 666		
15	Concrete, cement, sand and gravel, 4 months old; 1 bbl. Portland cement to each cu. yd concrete. Piers 3 by 2.81 by 3.85 in. high.	9	84 Cracked	Gradually crushed to pieces under continuous load of 45 tons					This test was made at the Dickson Works of the Allis-Chalmers Co. in this city, by William Griffith. The proportion was 1 of cement to about 7 of gravel and sand

TABLE II.—*Value of Various Artificial Roof-Supports.*

Kind of Material Comprising the Artificial Supports	Approximate Depth in Feet of Column of Coal-Measure Rock 1 Foot Square, Necessary to Compress Artificial Roof-supports						Remarks.
Per Cent of Settlement,	1 Per Cent	3 Per Cent	5 Per Cent.	10 Per Cent	20 Per Cent	30 Per Cent	
1. Rectangular gob-piers, ordinary construction,		10 ft	18 ft	36 ft	125 ft	306 ft	
2. Circular piers of mine-rock, well constructed,		16 ft	75 ft.	116 ft	292 ft	512 ft	
3 Timber cogs filled with gob, average construction,		8 ft	182 ft	228 ft	270 ft	419 ft.	Free to expand laterally.
4 Loose pile of broken sandstone through 1.75-in ring, 40 per cent. voids,			20 ft	53 ft	124 ft.	298 ft	
5. Pile broken sandstone, 40 per cent voids, voids filled with sand,			21 ft	53 ft	186 ft	165 ft	
6 Loose pile large size broken sand rock, 45 per cent. voids,		48 ft	66 ft	121 ft	351 ft	192 ft	
7 Mine-room filled with large broken sand rock, 50 per cent voids,	12 ft	27 ft.	45 ft	117 ft	131 ft	27 per cent settlement 615 ft	
8 Mine-room filled with broken sandstone, 40 per cent voids,		41 ft	71 ft	177 ft	619 ft.	1,310 ft	
9. Mine-room filled with broken sandstone, 40 per cent voids filled with sand,		46 ft	77 ft	325 ft	6,000 ft	23 per cent settlement 8,860 ft	
10. Mine-chamber filled with dry coal-ashes, 64 per cent voids,		13 ft	25 ft	70 ft	143 ft	332 ft	
11. Mine-room filled with dry river-sand,	12 ft	40 ft.	70 ft	142 ft	1,715 ft	6,640 ft	
12 Mine-room filled with river-sand, flushed in with water,	111 ft	522 ft.	891 ft.	2,310 ft		20¾ per cent settlement 8,860 ft	
13 Mine-chamber filled with coal-culm, flushed with water,	32 ft	118 ft	190 ft	472 ft	1,822 ft.	5,905 ft	
14. Concrete pier, 1 cement, 7 sand and gravel, 5 months old,	117 ft	1,092 ft	Gradually cracked to pieces under continuous load equal to 600 ft of rock				
Resistance of flushed culm,	1	1	1	1	1		Comparative
Resistance of flushed sand,	3 5	4 4	4 7	5	4		
Concrete pier,	3 6	9	Worthless.	Worthless	Worthless		

TABLE III.—*Horizontal Area in Square Yards of Artificial Mine-Pillars of Confined Flushed Culm or Flushed Sand Required Under Various Permissible Settlings to Sustain One-Third of the Over-Burden of One City Block of* 5 *Acres, at Various Depths.*

Ultimate Uniform Settling Permitted	Depth 25 Ft. Culm	Sand	Depth 50 Ft Culm.	Sand.	Depth 100 Ft Culm	Sand.
Per Cent.						
3	3,424	800	6,848	1,600	13,696	3,200
5	2,122	452	4,244	904	8,488	1,808
10	848	176	1,696	352	3,392	704
	Depth 200 Ft.		Depth 400 Ft.		Depth 800 Ft.	
3	Openings filled	6,400	Openings filled	12,800	Openings filled	Openings filled.
5	16,976	3,616	Filled	7,232	Filled	14,464
10	6,784	1,408	13,569	2,816	Filled	5,632

NOTES.

1. Up to 3 per cent. compression, piers of sand-and-gravel concrete might be only one-half the size of sand piers, but for weight which would produce greater compression they are worthless.

2. One city block of 5 acres covers 24,200 sq. yd.

3. In fixing upon the amount of settlement that might be permitted, consideration should be given to the fact that where there are several beds to be filled the total settlement will be several times as great as for one seam of the average thickness.

4. It will be noted that complete culm filling is necessary for settlement mentioned at from 200 to about 500 ft. depth of vein, while for greater depths the settlement, due to the greater weight, would be excessive; but sand, on account of its greater strength, is suitable for filling of all beds at all depths under the city of Scranton, and is therefore to be preferred.

TABLE IV.—*Approximate Cost Per Foot of Coal-Bed Thickness of Artificial Mine-Pillar of Confined Flushed Culm or Flushed Sand Required Under Various Permissible Settlings to Sustain One-Third of the Over-Burden of One City Block of* 5 *Acres, at Various Depths.*

Ultimate Uniform Settling Permitted.	Depth 25 Ft. Culm.	Sand.	Depth 50 Ft. Culm.	Sand	Depth 100 Ft. Culm.	Sand.
Per Cent.						
3	$286	$266	$572	$532	$1,144	$1,064
5	176	150	352	300	704	600
10	70	60	140	120	280	240
	Depth 200 Ft. Filled.		Depth 400 Ft.		Depth 800 Ft. Filled.	
3	$2,015	$2,128	Filled	$4,256	Filled	$8,069=$8,070
5	1,408	1,200	Filled	2,400	Filled	4,800
10	560	480	$1,120	960	Filled	1,920

The Preparation of Anthracite.

BY PAUL STERLING, WILKES-BARRE, PA.

(Wilkes-Barre Meeting, June, 1911.)

I. Introduction.

The general impression regarding the preparation of merchantable anthracite is that it is confined to a colossal, grimy structure, called a "coal-breaker." This name is a misnomer; for the desired result is not to break the coal, but to prevent its being broken.

Preparation may be said to begin at the face of the chamber, with the mining and loading of the coal. Local conditions vary, not only in the same field or basin, but also in the same mine, so that there is no fixed empirical rule governing the method of blasting or cutting coal. Tests should be made, when possible, to determine the explosive, or the mechanical coal-cutter, which will produce the largest percentage of what are locally known as "lump" and "prepared sizes" of anthracite. The prepared sizes are those mostly consumed for domestic purposes. All other sizes might be called by-products of the anthracite industry; for they are not especially desirable, being the degradation resulting from the mining and handling of a brittle or laminated material.

Being low in volatile combustible matter, anthracite burns most successfully when nearly of a uniform size, permitting the easy passage of air through the voids. This accounts for the large number of sizes into which the coal is separated.

Table I. gives the various sizes, the diameter of ring over and through which each size is made, and the usual purpose for which it is employed.

TABLE I.—*Commercial Sizes of Anthracite.*

Name	Diameter of Ring.		Use
	Over	Through	
	Inches.	Inches.	
Lump........	$6\frac{1}{2}$	....	Locomotive steam-coal.
Steamboat...	$4\frac{1}{2}$	$6\frac{1}{2}$	Blast-furnaces; smiths' forges.
Broken... . ..	$3\frac{1}{4}$	$4\frac{1}{2}$	Domestic furnace-coal.
Egg...........	$2\frac{5}{16}$	$3\frac{1}{4}$	Domestic furnace-coal
Stove...	$1\frac{5}{8}$	$2\frac{5}{16}$	Domestic range-coal
Nut.......	$\frac{13}{16}$	$1\frac{5}{8}$	Domestic range-coal
Pea.	$\frac{5}{8}$	$\frac{15}{16}$	Domestic furnace-coal
Buckwheat. .	$\frac{7}{16}$	$\frac{5}{8}$	Boiler, steam.
Rice...	$\frac{1}{4}$	$\frac{7}{16}$	Boiler, steam.
Barley..	$\frac{3}{16}$	$\frac{1}{4}$	Boiler, steam.

The coal, after being mined, is loaded either by hand (in flat workings) or from loading-chutes (in pitching veins). The former method does not seriously increase the breakage, while the latter does, and also contributes to the decrease in prepared sizes at the mines where it is employed. Hand-loading permits the removal of most of the impurities, such as rock or slate, and sends cars of fairly-clean coal to the breaker for further preparation, while chute-loading draws all the material mined into the car, and usually sends out a highly-impure product.

Where hand-loading is practiced in fairly-clean veins, the tonnage of chestnut and larger sizes shipped may be as high as 2.3 tons per 100 cu. ft. of mine-car capacity; while in other mines, with chute-loading and a very dirty run-of-mine product, it may be as low as 1.2 tons, and the amount of all sizes may vary from 2.7 to 1.5 tons per 100 cu. ft., respectively.

In the same region, under similar conditions, with a good run-of-mine, the product of prepared sizes may vary from 1.75 to 2.3 tons per 100 cu. ft. of mine-car capacity. The difference may be attributed to the varying conditions of the coal-beds themselves; to losses occasioned by jars due to running over uneven and poorly-constructed roads; to frequent dumping; to severe bumping of cars during motor-haulage; and, in the breaker, to poorly-constructed dumps, high drops of coal, long running-chutes with abrupt turns, and poor types of rolls used in crushing.

Under the well-known conditions of the anthracite-field, the

general methods of preparation may be summarized under three classes, namely, (I.) dry preparation; (II.) dry and wet preparation; and (III.) wet preparation; of which the one to be adopted depends on the quality of coal to be mined.

Class I. (Fig. 1) is employed when the seams of coal mined are dry, or are practically free from impurities, or where

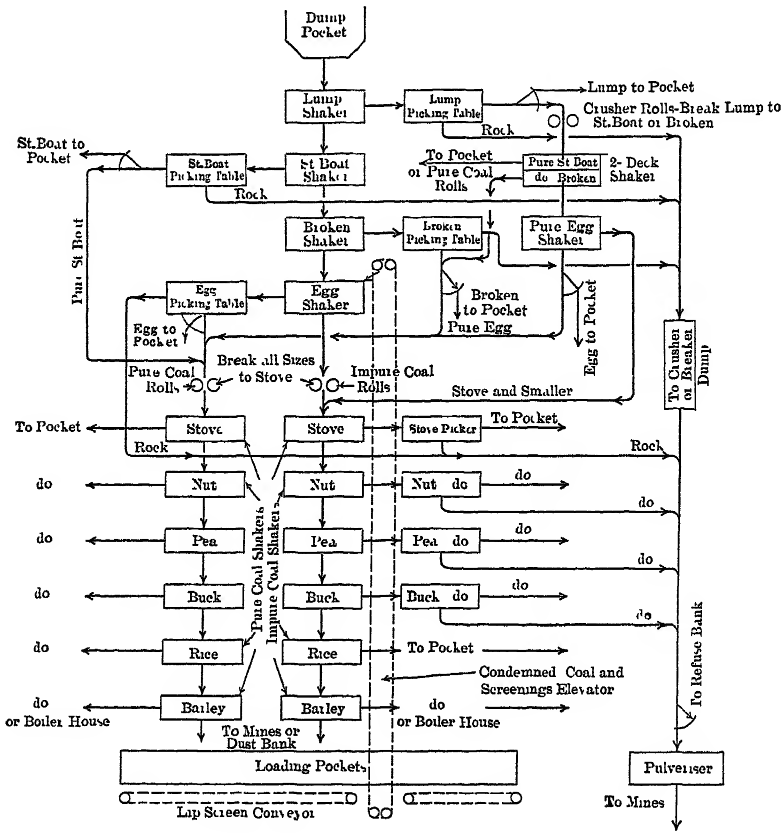

FIG 1.—PREPARATION-DIAGRAM, SHOWING TYPICAL RUN OF COAL DURING DRY PREPARATION.

the benches of slate occurring in the seams cleave free from the coal, and may be removed during hand-loading, and the run-of-mine contains generally not over 7 or 8 per cent. of rock or slate, which may be removed by hand-picking or by dry mechanical separators.

Class II. (Fig. 2) is employed when the run-of-mine contains a high percentage of impurities, including rock, slate, and bone. This percentage may be as high as 55 per cent., but the run-of-mine must contain large lumps of pure coal, which can be handled as a separate product, as in the first class. The sizes smaller than lump are sized and cleaned, using water to wash the product, to improve its appearance, and to remove the impurities by jigging.

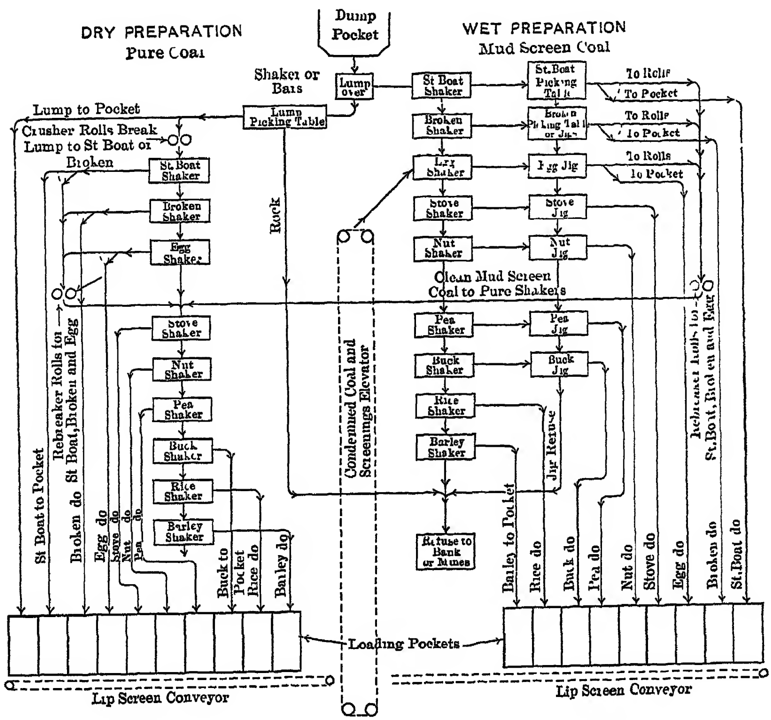

FIG. 2.—PREPARATION-DIAGRAM, SHOWING TYPICAL RUN OF COAL DURING WET AND DRY PREPARATION.

Class III. (Fig. 3) is adopted when the run-of-mine is high in impurities and shows a discoloration, as is the case near the outcrop of the vein, or when the entire product comes from wet, dirty seams, requiring a thorough washing to remove the dirt and discoloration.

Class I. presents the ideal breaker, with the advantages of low costs of installation, operation, and maintenance. More-

over, shipments of dry coal are very desirable to the trade, as free from the risk of the freezing of coal in cars, and the subsequent trouble of unloading it.

Class II. retains to some extent the advantage of dry coal-

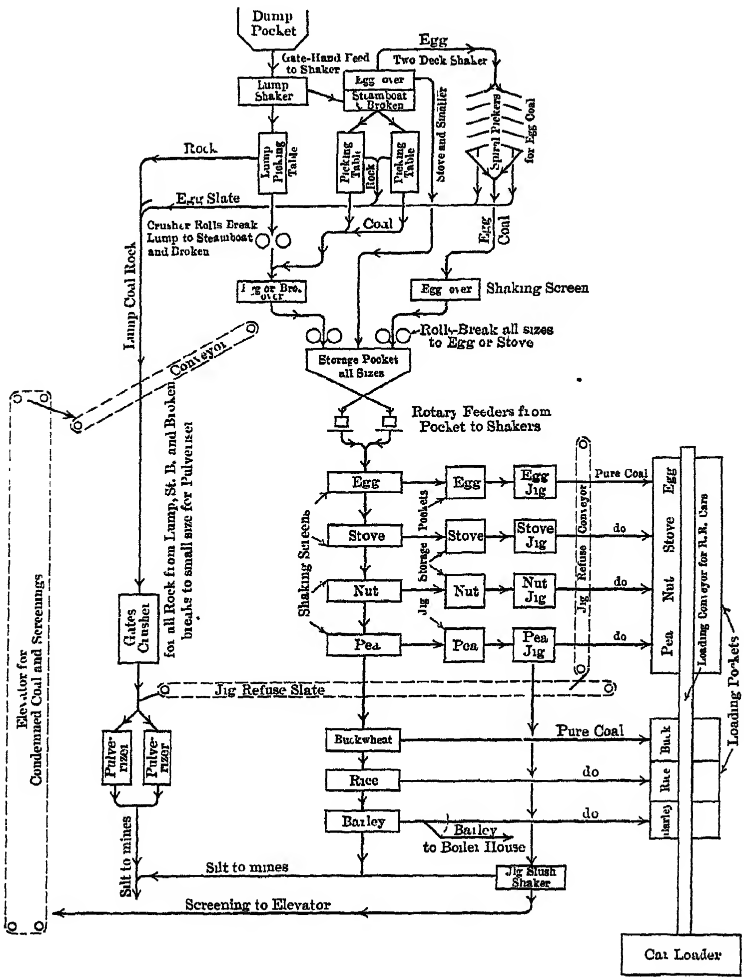

FIG. 3.—PREPARATION-DIAGRAM, SHOWING TYPICAL RUN OF COAL DURING WET PREPARATION.

shipments, but is higher in first-cost, operation, and maintenance than Class I. or III.

Class III. permits no dry shipments, and is higher in first-cost, operation, and maintenance than Class I.

In breaking anthracite from a larger to a smaller size, there is not, to my knowledge, a roll or crusher that will produce at will a fixed proportion of any one of the smaller sizes, and hence it is not possible to pre-arrange shipments with any degree of accuracy, so as to meet the demands of trade. This results, at times, in an overproduction of certain sizes, which must be shipped to storage-plants and stored until the market-conditions change, when this coal is reloaded and reshipped.

Table II. shows the proportions of various sizes produced in average practice at anthracite-breakers of the first two classes.

TABLE II.—*Product of Commercial Sizes in Breakers.*

Class I		Class II.		
Hand-Loading and a Clean Run-of-Mine Coal.		Chute-Loading and a Dirty Run-of-Mine, Containing Slate and Bone.		Hand-Loading and a Clean Run-of-Mine Coal.
		When Shipping Steamboat.	When Breaking Down Steamboat.	
	Per Ct	Per Cent.	Per Cent.	Per Cent
Lump and Steamboat...	11.19	5 56		
Broken........	12.41	7.05	1.36	10 07
Egg	14.68	13.88	7.11	10.96
Stove.......................	13.59	14.26	31.04	21.38
Nut.........................	26.64	21.00	30.05	27.78
Pea	6.37	10 31	14.24	9.82
Buckwheat	7.77	18.06	16.17	8.35
Rice	4.80	6.54	0 03	6.80
Barley..	2.55	3.34	....	4.84
Average price	$2.65	$2.43	$2.83	$2.56

The bottom line in the table shows what would be the average price per ton received for all sizes shipped, with the percentage of shipments shown in each column, and at the established prices for anthracite f. o. b. cars at the mines. The advantage of high mine-car yield in the prepared sizes, and the importance of eliminating breakage, which creates losses during the course of preparation, either in the mine or in the breaker, are shown by these comparative figures.

Fresh-mined coal is elevated to the "head" of the breaker and dumped into a large hopper or dump-chute, from which it is fed, by either a hand-operated gate or a mechanical feeder, over a sizing-screen, or a set of stationary or oscillating bars, which removes the lump-coal, allowing the smaller sizes

to fall through for further sizing and cleaning. The undersize is usually termed the "mud-screen" product. The lump-coal is inspected on a moving picking-band or table, or in an inclined gravity-chute, and the pure rock or slate removed by hand. Pieces of rock to which coal is adhering are placed on a special table, where the pure coal is chipped loose and returned to the table; the rock going to a rock-chute. Pieces of doubtful or "bone"-coal are also removed and sent to be prepared with the mud-screen coal. All coal that has passed over the picking-head or platform is free from impurities and is termed the "pure coal" product, which is delivered after inspection either into a lump-coal storage-pocket for shipment, or into a set of crusher-rolls to be broken into smaller sizes. The No. 1 or crusher-rolls break the lump into steamboat or broken and smaller sizes. If there is no sale for steamboat, broken, or egg, these classes, after sizing over screens, are passed through a set of rolls, which break them to stove and smaller sizes. The entire pure-coal product is now screened into its various sizes and stored in pockets, ready for shipment.

The mud-screen product is carried over a second set of screens, which size out steamboat and broken coal. These two sizes are cleaned of impurities either by hand-picking on a stationary or movable table, or by mechanical means, and are then either shipped, or re-broken into smaller sizes. In the latter event, the cleaned steamboat or broken coal is mixed with the pure-coal product and prepared with it. The mud-screen product falling through the "broken"-screen is separated into the various sizes, each of which is treated for the removal of impurities before going to the storage- and loading-pockets.

The method of cleaning varies from the hand-picking of the larger sizes, to a mechanical separator, operating on the difference in the coefficient of friction between the coal and its impurities when sliding over a smooth surface, or the jig, the theory of which is based on the different specific gravities of the minerals.

After sizing and cleaning, the coal runs to the storage-pockets. In loading it afterwards into cars for shipment, the fine dust and screenings (generally made in gravitating the coal in chutes from the screens into the pockets, or by mechanical handling) are removed by passing the coal over a punched

steel plate or a woven-wire segment, called a lip-screen. The lip-screen product is elevated to sizing-screens, re-sized and returned to the pockets. After loading, the coal is again inspected to make sure that it will pass a standard-test for size and purity. If it fails in this inspection, it is condemned, unloaded and re-prepared. It is, therefore, necessary in designing the breaker, to provide machinery for treating the condemned coal.

All the breaker-refuse of slate, bone, or rock, removed during preparation, is carried to a central point for final disposition. Two general methods are employed for this purpose: (1) The waste is deposited on the surface in banks, by means of a chain- or belt-conveyor, or by means of dump-cars, with mule or mechanical haulage; or (2) it is returned into the mines to fill the openings left by the extraction of the coal. The latter method, called "silting," is accomplished by crushing the refuse to a size which will pass through a 1.75-in. ring, and hydraulicking it into the mines through wooden or metallic pipes.

The disposition of refuse on the surface by means of a conveying-system is to be recommended when dumping-ground is available adjacent to the breaker. Banks can be carried to a height of 100 ft. or more, and extended in length by means of additional horizontal conveyor-lines. The use of the dump-car is advocated when the breaker is not tributary to the refuse-bank; but this method is more expensive in operation, and generally in first-cost, than the former.

The second method, "silting," is employed when dumping-room on the surface is limited or unavailable, and has the advantage that it adds to the stability of the remaining pillars and helps to support the roof of the mine. It is simple and cheap in operation, providing there is ample water-supply at very low cost; but a complete installation is more expensive than that of a conveying-system, if the cost of the pump and appurtenances, necessary to rehandle the silt-water from the mines to the surface, is included.

Table III. gives a standard of preparation which is about the average adopted in the anthracite coal-field. The table allows a percentage of "bone," in addition to slate, in the coal;

"bone" being defined as a product containing between 40 and 55 per cent. of carbon.

TABLE III.—*Standard of Preparation, Showing the Percentage of Slate, Bone, etc., Permitted in Each Size of Coal.*

May Contain	Broken.	Egg	Stove	Nut.	Pea	Buckwheat	Rice.	Barley.
Of slate....	1	2	2.5	4	8	10	15	15
Of bone....	2	2	4	5	5			
Of next size larger............	...	5	5	10	5	8	8	8
Of next size smaller... ...	20	50	50	15	15 B 15 R.	15	25	

II. MACHINERY.

All breaker-machinery should be simple in construction, so that it can be operated and maintained by the ordinary workman, without the requirement of a force of expert machinists to make repairs. It should be reliable in accomplishing results, without the constant attention of an attendant. And it should have interchangeable parts, so that a large supply of pieces for repair need not be maintained at great expense. Moreover, it should be as nearly "fool-proof" as possible. Such machinery is subject to severe and heavy shocks, to excessive wear, to vibration, and generally, in wet preparation, to the action of the water pumped from the mines, which often contains as much as 160 grains of free sulphuric acid to the gallon. In wooden breakers, shafting and drives get out of alignment through the uneven settlement of the timbers, bringing tremendous strains on the machinery, and increasing the power required to drive it. This difficulty may be overcome by more rigid construction, either of steel or of reinforced concrete, or a combination of both. Fig. 4 is the side-elevation of a timber breaker; and Fig. 5 is a section of the loading-pockets of the same breaker.

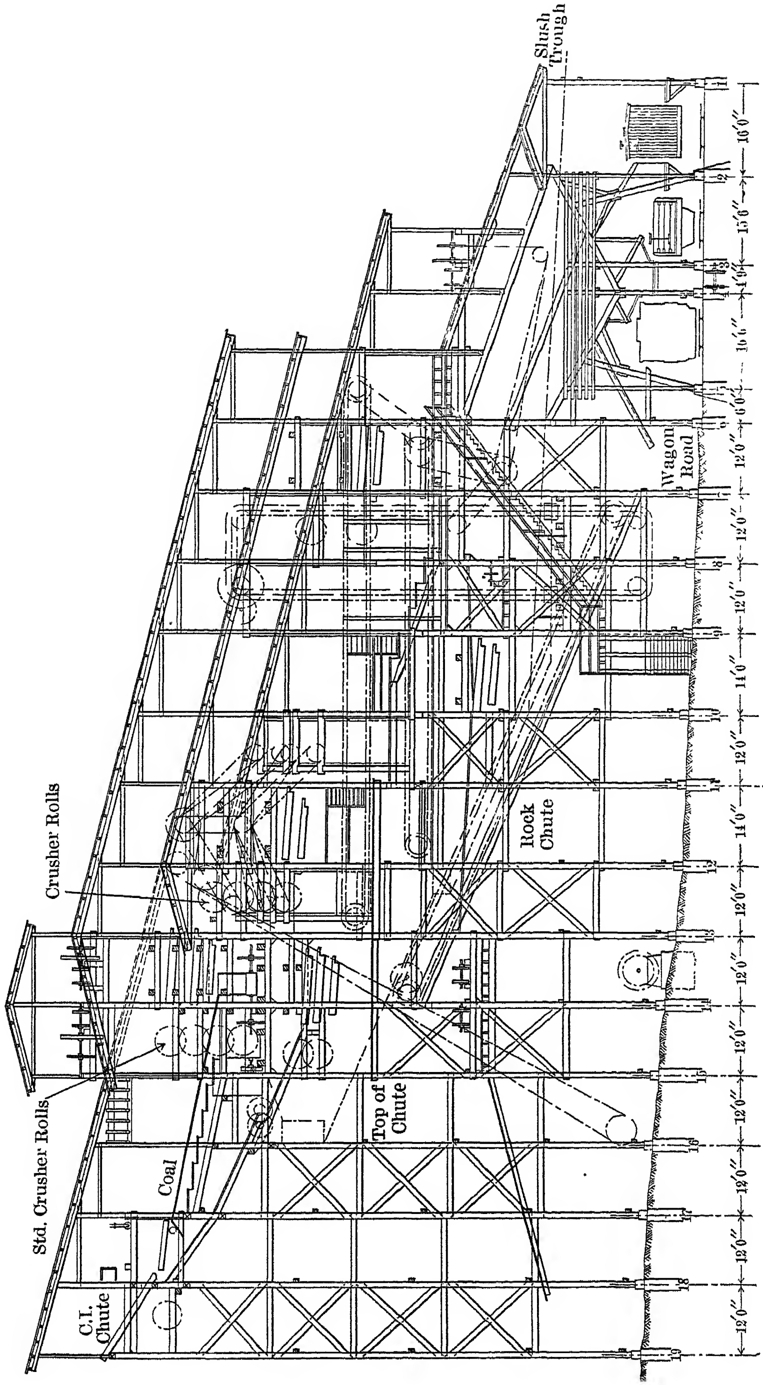

FIG. 4.—SIDE-ELEVATION OF TIMBER-CONSTRUCTED BREAKER.

1. *Sizing-Machinery.*

This consists of screens, classified as shown in Table IV.

TABLE IV.—*Classification of Screens.*

FIXED.	MOVABLE.
Adjustable-bar screens.	Cylinder- or revolving-screens.
Finger-bar screens.	Shaking-screens.
Punched steel-plate screens	Gyrating-screens.
Woven-wire segment screens.	Oscillating- or movable-bar screens.

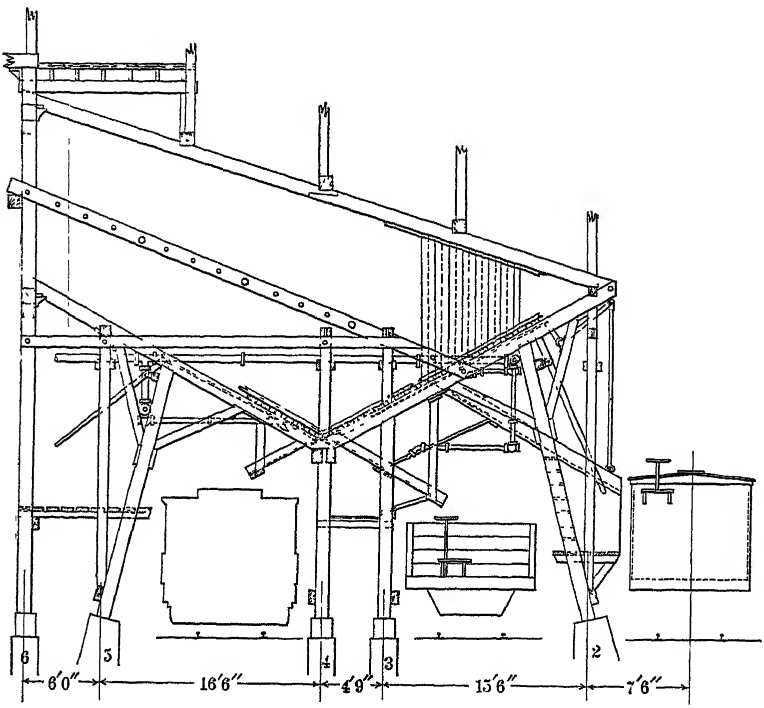

FIG. 5.—SECTION THROUGH LOADING-POCKETS OF BREAKER SHOWN IN FIG. 4.

a. Fixed Screens.—The fixed screens are usually built into the chute, and on a suitable pitch, down which the coal will slide, allowing the smaller sizes to pass between the bars, while the larger pass over. They are generally used, not when uniform sizing is required, but for a preliminary separation of the larger sizes from the smaller, before cleaning, and to remove the dust and fine chippings made during preparation.

The adjustable-, finger-, and oscillating-bar screens have openings of which the ratio of length to breadth varies from about 10 : 1 to 20 : 1, allowing the passage of flat pieces of coal, together with the more cubical pieces. In other types of screens, the opening is either round or square, and the sizing resulting from their use is more exact. The latter types are recommended for the final sizing of coal.

The adjustable-bar screens, Figs. 6 and 7, are often used just below the dump-hopper. The bars are spaced from 6 to 7 in. in the clear, allowing lump-coal to pass over and the smaller sizes to fall through. In Fig. 6 the bar K-28 dove-tails into a V-groove in the upper and lower bar-rests, K-25, K-26, and K-27. These V-grooves are continuous at about 1-in. pitch per foot, so that the opening may be increased or decreased in increments of 1 in., in order to adjust the quantity and size of coal going to the picking-table.

The bar has a pitch of 4 in. per foot, allowing the coal to slide over, and the lower end is open, to pass any pieces which, hanging between the bars, would otherwise jam at the lower end and require frequent cleaning. The top flange of the bar is also tapered in its length; and the bar-rests hold the center-lines of the bars parallel, while the tapered side of the flange gives a V-shaped opening between the bars, wider at the lower end than at the top, which also allows the coal to free itself, and prevents blocking.

The finger-bar screen is not essentially a sizing-screen, being designed more especially to remove from the prepared coal pieces of flat slate or bone, when their percentage is so high as to condemn the product on inspection. It is often constructed of round bar-steel, placed in the same horizontal plane, and on centers which vary with the size of coal from which the flat material is to be screened. The lower ends of alternate bars are bent down below the plane of the intermediate ones, while the upper ends still remain in the same plane. This deformation gives a V-shaped slot, as in the adjustable bar screen, and prevents blocking. Another type is built up from angle-iron, with the legs looking down at an angle of 45°. The top edge of the angle is maintained in the same plane and the V-slot is made by planing a taper edge on the outer legs of the angle. A similar screen is made of cast-iron at much less cost.

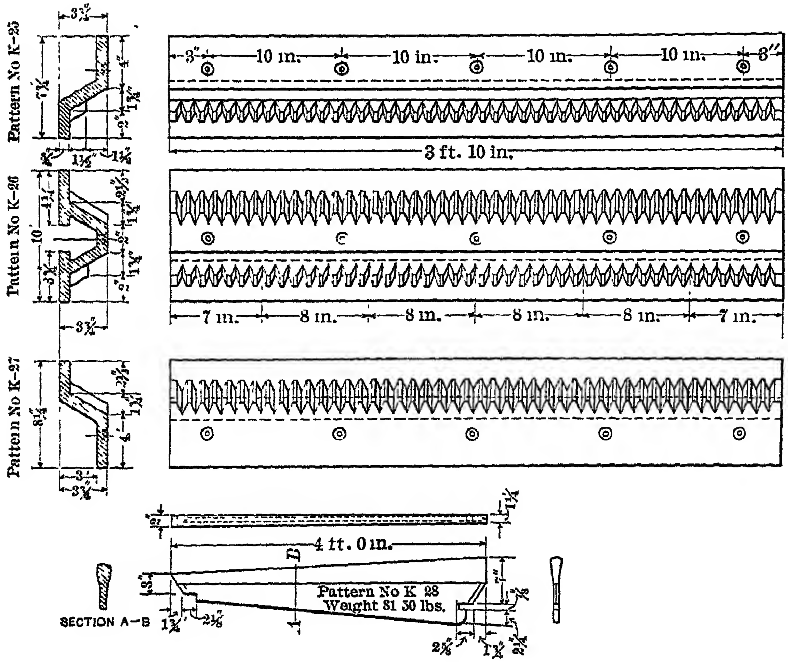

FIG. 6.—DETAILS OF CAST-IRON ADJUSTABLE-BAR SCREEN.

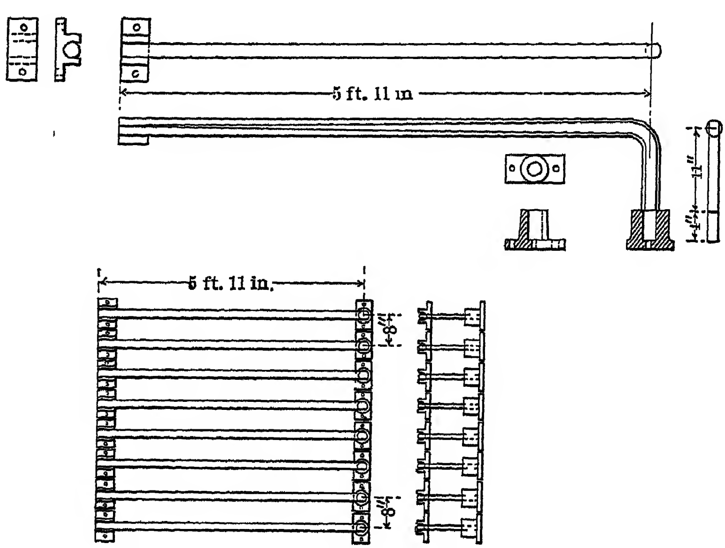

FIG. 7.—STEEL ADJUSTABLE-BAR SCREEN.

The punched steel plate and the woven-wire segment are cheap and convenient devices employed to remove dust and small chippings. They are placed in a chute, and the coal passes over them, dropping out the finer material. The punched plate for "lip-screens" has been already mentioned. In such screens, the ratio of width to length of opening is 1 to 2.

The advantages of fixed screens are: (1) they are inexpensive in first-cost; (2) they require no power for operation; (3) they need practically no attention, except when blocked; and (4) their capacity is large.

Their disadvantages are: (1) they effect a poor sizing of the coal; (2) being set with a pitch, they require additional height of breaker (except the finger-bar, which may be adjusted to a shaking-screen without increasing height); (3) the adjustable bar involves a drop at the lower end, which increases the breakage and consequent loss of prepared sizes.

b. Movable Screens.—The revolving- or cylinder-screens are usually from 6 to 8 ft. in diameter, and consist essentially of a central shaft, supported at the ends by boxes, in which it rotates, and carrying spiders, to the outer end of the arms of which is bolted a ring, to which the screen-jacket of woven wire-mesh is attached. The coal enters through a head-wheel, with teeth cast on the circumference. A pinion engaging with this gear rotates the screen. The maximum periphery-speed should not exceed 250 ft. per minute. The screen-shaft is usually placed on a pitch of 0.75 in. per foot, and the revolution of the screen moves the coal forward from the head-wheel to the discharge-end.

In the breaker, this type is giving way to shaking-screens; but it is better adapted than the latter to storage-plants, where it is not always possible to wash the coal in reloading. The revolving-screen causes the coal to roll over itself, the action being similar to that of a cleaning-mill, as used in a foundry to remove sand from castings.

Its advantages are: (1) it effects an exact screening and sizing of the coal; (2) revolving at a slow speed, it does not tend to vibrate the breaker-structure, as is the case with the shaking-screens.

On the other hand, it presents the following disadvantages: (1) high first-cost and maintenance, as compared with shaking-

screens under similar conditions; (2) small capacity; (3) requirement of more space than shaking-screens of the same capacity; and (4) the fact that only about one-eighth of the screening-surface is in contact with the coal at one time.

The shaking-screen, or "shaker," consists of a rectangular box-like framework (Fig. 8), with sides of steel plate or timber, connected by cross-angles, to which sides the sizing-jackets of punched steel plate are bolted. The entire machine is hung from its supports, or bridge-trees, by means of flexible chains, rigid hangers with pin-connected ends, or oak or hickory boards with fixed ends.

Rectilinear motion is obtained from a pair of eccentrics, located on a driving-shaft and connected to the shaker by wood or steel eccentric-rods or arms. The shaker-end of the eccentric-rod is connected to a wrist-pin, usually located at or near the center of gravity of the shaker, or often bolted rigidly to the shaker-side. When wooden arms are used the arm is reduced in section at its center to make it flexible, so that it can accommodate itself to the change from rotary to rectangular motion of the eccentric and shaker, respectively.

The center of gravity moves in the arc of a circle, the chord-length of which equals the eccentric-travel, while its radius equals the vertical distance from the suspension-point to the center of gravity. The shaker is so located relative to its hangers that, at mid-travel of the eccentric, the shaker-hangers are vertical, and the shaker is at the lowest position in its arc of travel—reaching its highest position twice in a revolution of the driving-shaft, *i.e.*, at each end of the eccentric-travel. The shaker is inclined about 1 in. per foot in the direction of the flow of the coal, which travels downward by reason of the fact that the resultant of all forces acting on each piece of coal when the shaker is moving in a forward direction, is greater than the resultant from similar forces when the shaker is moving on its backward stroke. The upward motion of the shaker in its extreme position tends to throw each particle of coal up, and prevents it from blocking or hanging in the mesh. The best results are obtained when the ratio of length of hanger to eccentric-travel is from 4 : 1 to 15 : 1. When this ratio exceeds 15 : 1, the vertical force in extreme positions is of little benefit in keeping the mesh open.

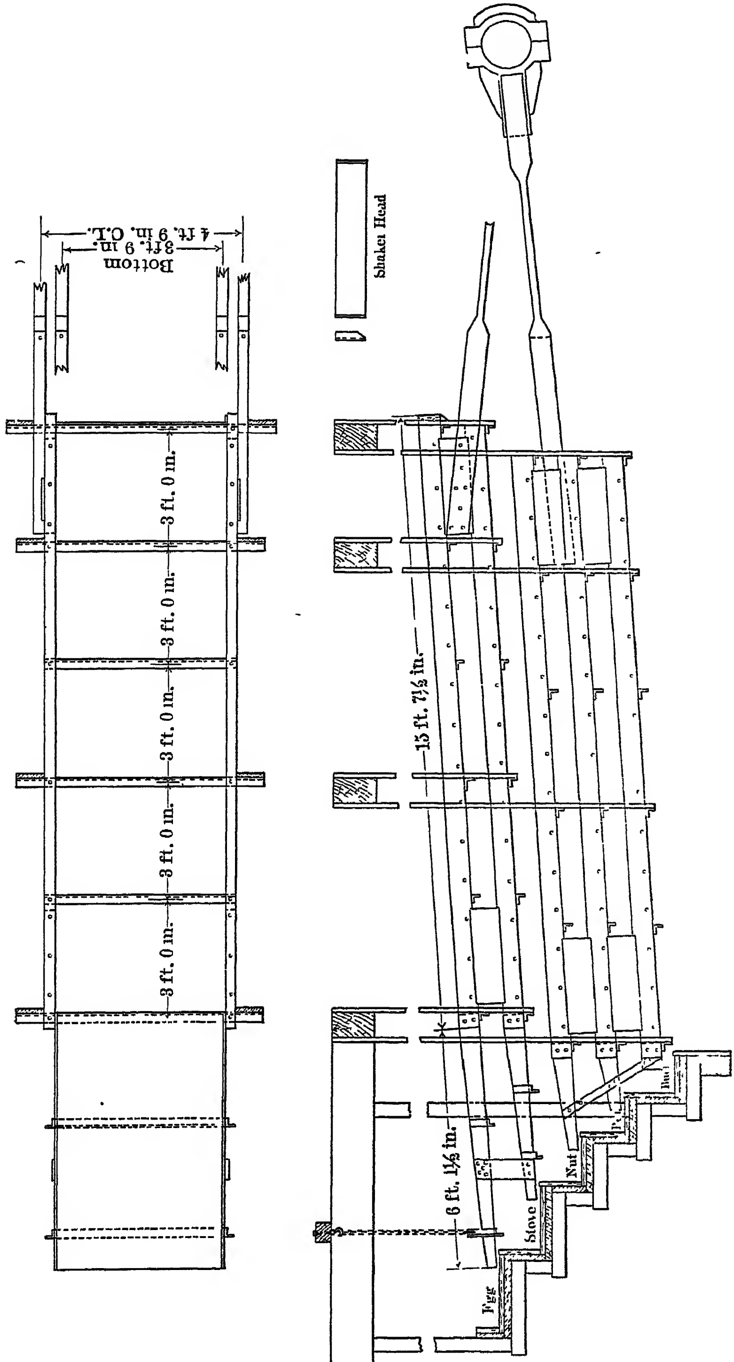

FIG. 8.—4- BY 15-FT. SHAKING-SCREENS ARRANGED TO SIZE FIVE DIFFERENT SIZES OF COAL.

The usual stroke of the eccentric is 6 in., and the speed of the driving-shaft is from 140 to 150 rev. per min. Recent experiments show that a stroke of 6 in. is better for the larger sizes of coal, and may be reduced for the smaller.

Shakers are usually built with one set of mesh- or sizing-plates. When it is desirable to make two or more sizes of coal on one shaker, it may have two or more sets of plates, one above the other, each set of plates allowing its respective size of coal to pass over, while the smaller sizes fall through.

On account of the excessive vibration set up by the shakers, it is best to hang them, not singly, but in pairs, with the driving-eccentrics 180° apart on the driving-shaft. When spring-board hangers with fixed ends are used, the elasticity of the hanger, which must be at least 15 times the travel in length, aids it to vibrate and tends to return it to a neutral position. This property of the wooden hanger with fixed ends produces a smoothly operating machine, and assists in eliminating vibration when running.

Woven-wire screens for shaker-jackets are not practicable, since the movement of the coal over the mesh soon wears the wires. Moreover, if mine-water is used to wash the coal, it oxidizes them; and finally, the vibration results in a change in the pitch of the wires, and an increase or decrease in the area of the mesh, and, consequently, imperfect sizing.

Punched plate has not this disadvantage. The square-punched hole allows closer spacing, and consequently greater area through which the coal may pass, but is more liable to block than the round mesh.

The advantages of shakers are: (1) exact sizing; (2) low first-cost; (3) accessibility for repairs and inspection; (4) simplicity of construction (the wooden style, with wooden eccentric-arms and hangers, may be built at the colliery with the ordinary force of mechanics); (5) reliability in operation; (6) saving in pitch, and thus in required height of breaker, as compared with adjustable-bar screens; (7) large capacity; (8) usefulness, at the head of the breaker, when the run-of-mine contains pieces of coal not exceeding 150 lb. in weight, to size coal going to the picking-room, since a uniform size of product is more easily cleaned and inspected than a poorly-sized mixture.

Their disadvantages are: (1) the vibration which, unless well balanced, they communicate to the breaker-structure; (2) their undesirability for storage-plants, already mentioned under "Revolving-Screens," unless water be used for washing.

Table V., giving the average number of square feet of shaker required per ton of coal treated in 10 hr., has been compiled from actual working-conditions. The area for steamboat- and broken-coal is made large, to receive the rush of coal from the dump. When a large dump-hopper is used to hold the coal, and a feeder to control the supply, this area can be greatly reduced. The area for egg-coal is made large, to receive the large amount produced in breaking down steamboat- and broken-coal in the rolls, and can likewise be reduced by installing pocket and feeders.

For a dump-shaker of 6.25-in. mesh, when perfect sizing is not required, the area needed per ton in 10 hr. is 0.05 square foot.

TABLE V.—*Area of Shaking Screen.*

Size	Mesh.	Sq. Ft. Per Ton in 10 Hours	
	Inches	Dry	Wet
Steamboat	$4\frac{1}{2}$		1.5
Broken	$3\frac{1}{4}$	0.60	1.2
Egg	$2\frac{5}{16}$	1.20	1.1
Stove	$1\frac{5}{8}$	0.25	0.35
Nut	$\frac{15}{16}$	0.20	0.27
Pea	$\frac{5}{8}$	0.61	0.69
Buckwheat	$\frac{7}{16}$	0.50	0.53
Rice	$\frac{1}{4}$	0.67	0.65
Barley	$\frac{3}{32}$		0.67

The gyrating- and oscillating-screens were thoroughly described in a paper by Eckley B. Coxe, at the New York meeting of the Institute, in September, 1890 (*Trans.*, xix., 398). They present the advantages of (1) correct sizing, and (2) large capacity; with the disadvantages of (1) high first-cost, as compared with shakers; (2) expense of repair and maintenance; (3) inaccessibility for inspection and repairs; (4) poor sizing of the smaller sizes, especially in wet breakers, where water is necessary to aid preparation, since the water can only be used on the top deck or mesh; and (5) impracticability of proper balancing, and consequently considerable vibration communicated to the breaker-structure.

The oscillating or movable bars are desirable for sizing the

lump-coal when the run-of-mine contains lumps weighing 150 lb. or more, since the design is suitable for severe work and heavy shocks. The driving-shaft usually runs 50 turns, giving 100 forward strokes per minute to the coal. This design of screen has the advantage of acting as a feeder under normal conditions; but when crowded from behind the coal is pushed over, carrying some of the smaller sizes with the lump, and its value as a feeder is lost. The space between the bars is fixed and cannot be changed without removal of the bars. The bars wear from the rubbing and sliding of the coal, and become thin in cross-section, allowing the bars to spring apart. This increases the distance between the bars, and allows some lump-coal to go with the mud-screen product.

Their advantages are: (1) the heavy construction, to handle large pieces of lump-coal; (2) the action as a feeder to the picking-table, under normal conditions; (3) the saving in pitch, and consequently in required height of breaker, as compared with adjustable bars; (4) the slow speed, not vibrating the breaker-structure.

Their disadvantages are: (1) poor sizing; (2) the fixed space between bars, which will not permit of adjustment; (3) the feeding-action is not reliable.

Table VI. gives the average number of square feet of oscillating-bars required per ton in 10 hr. These data have been compiled from the work of a breaker receiving fairly clean, hand-loaded, run-of-mine.

TABLE VI.—*Area of Oscillating-Bars and Lip-Screens.*

Oscillating-Bars.

Size	Width of Opening.	Sq. Ft. Per Ton in 10 Hours.
Lump.	6 in.	0.05

Lip-Screens, Using Punched Plate.

Size.	Mesh.	Sq. Ft. Per Ton in 10 Hours.
	Inches.	
Broken	2.0 by 4.0	
Egg	1.5 by 3.0	0.05
Stove	1.25 by 2.5	
Nut	0.75 by 1.5	0.75
Pea	0.50 by 1.0	1
Buckwheat	0.25 by 0.5	
Rice	0.125 by 0.375	2
Barley		

c. Rolls.—Coal is broken to smaller sizes either by hand, with sledges, picks and bars, or by rolls, or crushers.

The first method is employed only on the picking-head, either as an aid to cleaning, by breaking or chipping the slate loose from the coal, or in order to reduce large pieces of coal to a convenient size, which will enter the rolls. Elsewhere, rolls only are employed.

In general, the design is as follows: Twin drums (Fig. 9), varying in dimensions, into which are inserted hardened, pointed steel teeth. These drums are keyed to separate shafts, which revolve in boxes, supported on a cast-iron base-plate. The two drums are placed opposite and are protected by a cast-iron or steel-plate casing, with an opening in the top, through which the coal is fed. Cast-iron gears, one on each shaft, located outside the bed-plate, run together and cause the drums to revolve in opposite directions. The power is usually applied through a pulley, mounted on one of the shafts, which extends beyond the base-plate, and is supported at its outer end by an outboard-bearing. The tops of the rolls revolve towards each other, and the coal, being dropped into the top, is drawn through by the teeth and broken into smaller sizes. The pedestals supporting the driving-roll are fixed in position, being bolted to the base-plate. The driven roll is supported on an adjustable pedestal, which may be moved in or out in relation to the fixed roll, in order to increase or decrease the opening between the two rolls, and vary the proportion of sizes made in crushing. The limit of adjustment depends on the length of the teeth of the gears; and when a greater opening is desired than is possible with the gears in use, they are replaced by gears of a greater pitch-diameter. The adjustable pedestals are also equipped with a breaking-shell or cushion-spring. The former collapses when any foreign material, such as hard rock or steel, passes through the roll, while the latter is compressed, thus preventing the rolls or pedestals from being broken.

Some stress has been laid upon the design of roll-teeth, but it is my opinion that the style does not have much bearing on the results obtained, but that the greatest loss in prepared sizes in breaking is due to the points of the teeth overlapping, so that the coal, being caught on the point of one tooth, is

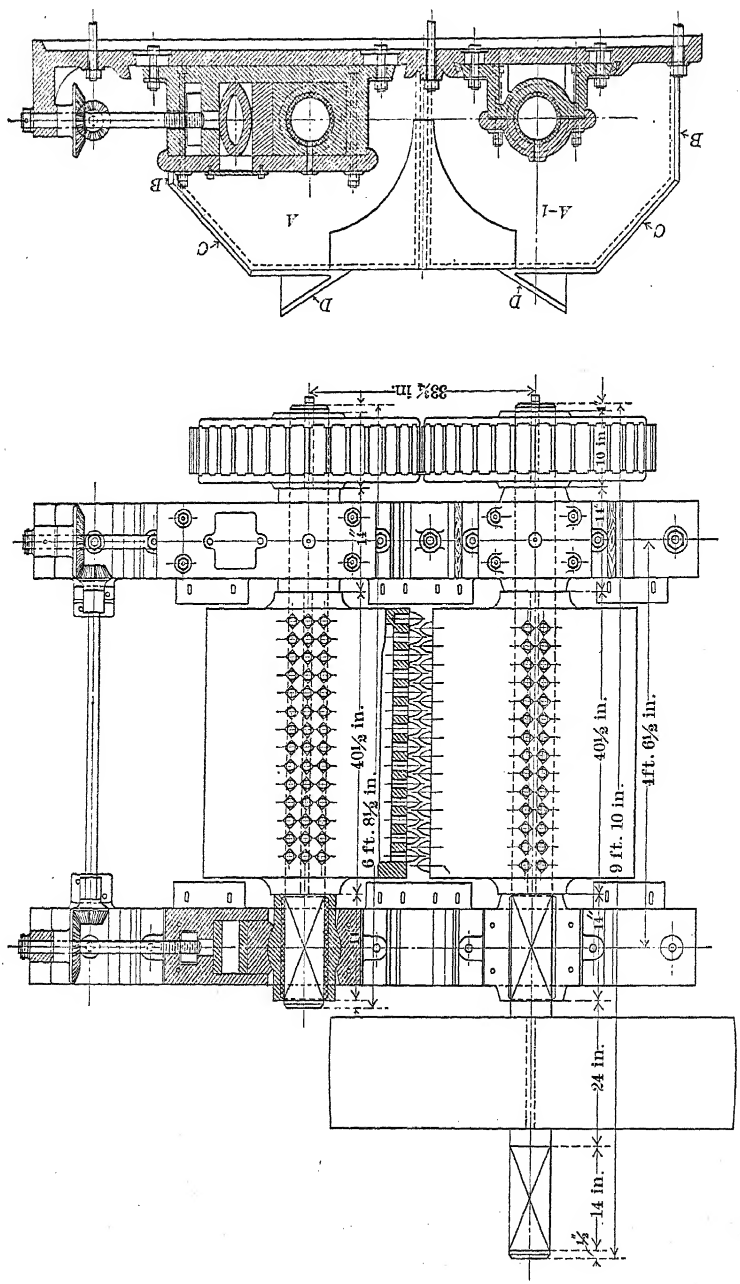

FIG. 9.—PLAN AND SIDE-ELEVATION OF A 30-IN. DIAMETER BY 36-IN. FACE ROLL, USED FOR BREAKING BROKEN AND EGG INTO STOVE AND SMALLER SIZES. PRESENT TYPE OF ROLLS, WITH OVERLAPPING TEETH.

crushed on the body of the opposite roll, while the adjacent teeth grind the broken pieces into small sizes or dust. If a large piece of coal, placed upon an anvil, receives a quick, hard blow with a pointed tool, such as a pick, breaking the lump and allowing the pieces to fly away, the percentage of pea and smaller sizes is very low, compared with the result of a similar test in which the coal is held on all sides, as would be the case in a roll with overlapping teeth. It is a fair inference that the style of tooth does not chiefly determine the amount of comminution, but that the location and length of the tooth have a serious bearing, and sharpness a preponderating influence. Moreover, tests on roll-speed have demonstrated that performance was dependent, first, on spacing and length of the teeth, and second, on peripheral speed.

Table VII. gives the usual sizes of bodies, pitch or spacing of teeth, size of teeth, and purpose of use, of various sizes of rolls in use in the anthracite coal-fields:

TABLE VII.—*Dimensions and Arrangement of Rolls and Teeth.*

Roll.	Size of Shell. Diameter	Size of Shell. Length.	Number of Teeth per Row.	Interval, Center to Center.	Number of Rows to Circle.	Size of Tooth.	For Crushing.
	In.	In.		Inches.		Inches	
No. 1...	51	41	8	4.5	36	3.5 by 1.75 sq	Lump to steamboat.
No. 1...	33.5	46	8	5.5	19	3.5 by 1.75 sq.	Lump to steamboat.
No. 2...	30	36	10	3.75	26	2.75 by 1.5	Steamboat to broken, or broken to egg.
No. 3...	30	36	14	2.42	39	2 by 1.25	Broken to egg, or egg to stove.
No. 6...	24	32 75	21	1 35	72	1 by 1	Egg-bone to stove. Stove-bone to nut or pea.
No. 6...	24	32.75	21	1.35	70	1 by 1.125	Nut-bone to pea.

Performance-tests of various rolls have usually shown, for a peripheral speed of about 900 ft. per minute, a loss in prepared sizes of from 15 to 30 per cent., when breaking lump-, steamboat-, and broken-coal to smaller sizes; and reduction of the periphery-speed has resulted in saving from about 4 to 15 per cent. in prepared sizes, as compared with high-speed tests.

But the ordinary direct-driven, high-speed roll could not be satisfactorily run at reduced speed, because the change involved a loss of crushing-power, causing the rolls to become choked with coal.

Table VIII. gives the results of tests, showing the size of roll, peripheral speed, size of coal fed to the roll, and percentage of the various sizes made.

TABLE VIII.—*Product of Rolls at Various Speeds.*

Number.	Size.	Speed	Size Fed.	Sizes Produced.									
				Broken.	Egg.	Stove	Nut.	Pea.	Buckwheat	Rice.	Barley.	Dirt.	Pea and Smaller
	Inches.	Ft. Per Min.		Per Cent.	Per Cent	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent	Per Cent.
2	30 by 36	942	Lump	48.6	14.0	8.9	9.4	4.6	5.2	2.5	3.0	3.8	19.1
3	30 by 36	1,037	Broken	...	42.3	24.1	13.6	4.8	5.5	2.3	3.0	4.4	20.0
2	30 by 36	173	Lump...	66.1	10.1	5.9	6.2	2.6	3.2	1.4	1.9	2.6	11.7
3	30 by 36	173	Broken	...	57.3	19.0	10.4	3.2	3.5	1.7	2.2	2.7	13.3
3	30 by 36	230	Broken	...	54.3	23.9	9.8	2.6	3.7	1.5	1.9	2.3	12.0

This table shows a saving in prepared sizes by running rolls at a slower periphery-speed than that which was generally adopted by pioneers in the preparation of anthracite coal.

It was also formerly recommended that sized coal only be fed to rolls to be broken down. This made it necessary to install a set of rolls for each size of coal to be broken, *i. e.*, one set to break lump to steamboat; one set to break steamboat to broken, etc. While this rule has not been completely followed in practice, the usual installation has included a set of No. 1 crusher-rolls, to break lump into steamboat and broken; one set of No. 2 or merchant-rolls, to reduce steamboat and broken to egg; one set of No. 3 or re-breaker rolls, to crack broken into egg and stove; or the three sets have been so adjusted that all the coal could be broken into stove and smaller sizes when there was no demand for the larger ones. The installation of these three sets of rolls materially increases the height of the breaker, and requires additional screens to separate the larger sizes from the mixture resulting at each operation. The smaller sizes, falling through the screens, pass on to the main sizing-screens.

The economical breaking of coal has been the subject of

much serious consideration; and the tests tabulated above have led to the design of a compound-geared slow-speed roll, not extensively used at present, with which I have had considerable experience. The object sought was an increase in prepared sizes, when breaking down from lump to egg or stove and smaller, with only two sets of rolls. On the basis of the knowledge that slow periphery-speed would give the best results, and with the use of teeth the points of which did not overlap, normally, the results shown in Table IX. were obtained.

TABLE IX.—*Results of Crushing at Low Speed.*

Size of Roll, 36 in. by 2 ft. 10 in. Periphery-speed, 250 ft per minute

Test No.	Size Fed.	Steamboat.	Broken.	Egg.	Stove.	Nut.	Pea.	Buckwheat.	Rice.	Barley.	Dirt	Pea and Smaller.
		Per Cent.	Per Cent	Per Cent	Per Cent.	Per Cent.	Per Cent.	Per Cent	Per Cent	Per Cent.	Per Cent	Per Cent.
1	Lump.. ...	9	49	12 8	9.4	7	3 3	4.4	1.4	1.4	2 3	12 8
2	Lump	6.8	41	15.3	11.2	8.7	4 3	5.3	2.4	2.4	2.8	17
3	Lump	64	14.1	3.8	5.7	4 3	2	2.6	1	1	1 5	8.1
4	Steamboat	16.7	59	9 8	4 7	3 5	1 6	2	0 8	0.8	1 1	6 3

The object of the first test was to determine the loss by double breakage. Lump-coal was fed to the roll, which was opened up to make about 60 per cent. of steamboat; the steamboat was screened out of the resulting mixture, the rolls were closed up, and the steamboat was broken to the next smaller size. In this test, the percentages given are the totals from both operations. In the second test, lump-coal was fed to the rolls set to break directly into broken, with a small percentage of steamboat. The results indicate the advantage of double breakage or of feeding coal of uniform size to the rolls. The third and fourth tests show separate results; the rolls being spaced for No. 3 as in the first half of No. 1, and for No. 4 as in the second half of No. 1. These results are similar to those shown in Table VIII., and confirm the conclusion that low speed is an important factor in economic roll-operation. The design of the teeth is of some importance; but additional experiments must be made before the determination of a form which will further increase the percentage of prepared sizes.

The compound-geared slow-speed roll (Fig. 10), used in the above test, is, as far as I have been able to learn, the first of

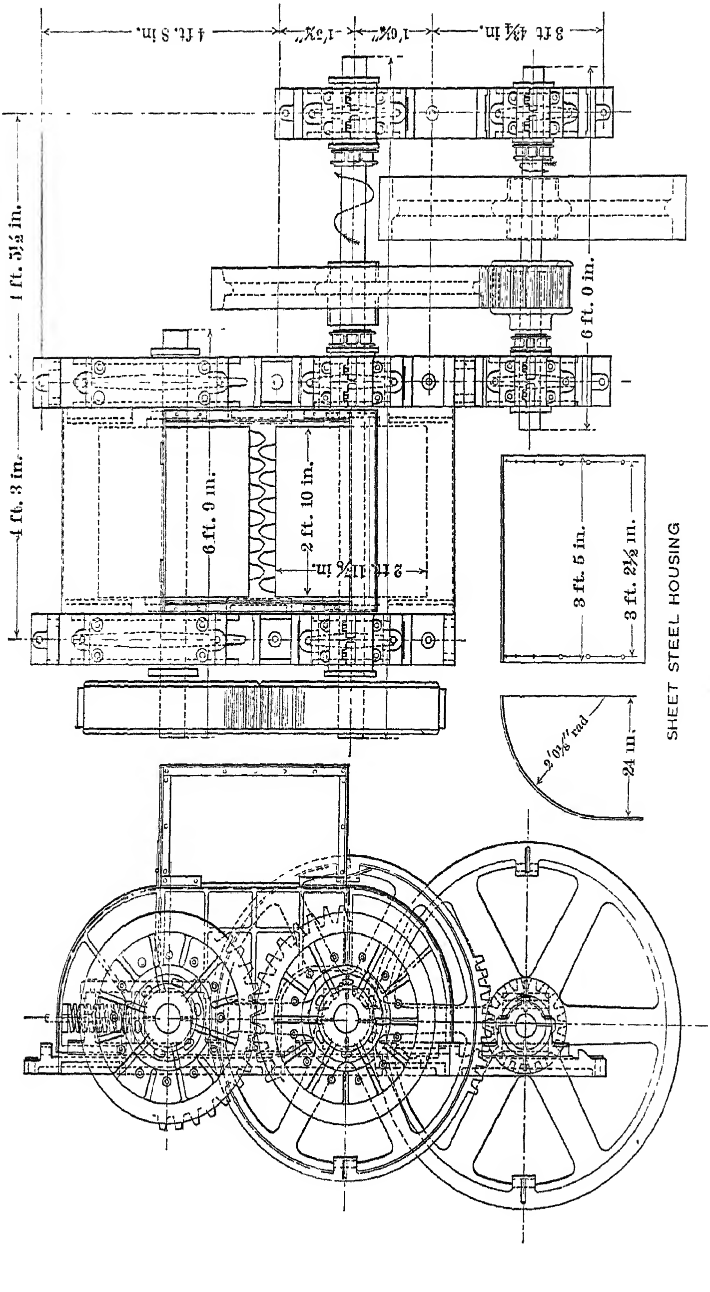

FIG. 10.—PLAN AND SIDE-ELEVATION OF 3-FT. BY 2-FT. 10-IN. COMPOUND-GEARED ROLL. DRAWING SHOWS OVERLAPPING TEETH, WHICH HAVE BEEN REPLACED BY A DIFFERENT STYLE OF TEETH, DESIGNED FOR BREAKING DOWN A DIFFERENT SIZE OF COAL.

this type in the anthracite-region. The teeth are cast in chills, as integral parts of a cast-iron segment, which is bolted to the roll-body. Their size and spacing vary according to the purpose required. The roll-body has eleven sides, each cast with a recess to receive a pad on the back of the segment. The body and segment are machined for a neat fit, and the latter is held in place by two bolts, one at each end. The pad takes all the shear and saves the bolts, which, in former rolls of this type, where the segment was not a dovetail-fit, sheared off, causing much damage and delay. The chilled cast-iron teeth wear as well as forged and tempered steel teeth, the first-cost of which is greater.

The fact that, by changing the segments, this roll may be used to break any size of coal, is a decided advantage, reducing the number of repair-parts to be kept on hand. The slow speed increases the duration; and the large fly-wheel pulley provides energy to handle any large lumps that tend to block the rolls. Spring-cushion pedestals are provided to take up shocks, and prevent breaking the rolls or bending the shaft, when any foreign material, such as hard rock or iron, passes the rolls.

Table X. gives the principal dimensions of the teeth used in the slow-speed roll-test, and adopted by me in practice.

TABLE X.—*Dimensions and Arrangement of Teeth.*

Size of Rolls, $35\frac{5}{8}$ in. by 2 ft. 10 in.

Type	Number of Segments to One Roll.	Number of Rows of Teeth per Segment.	Number of Teeth Per Row.	Interval.	Size.
				Inches.	
1	11	3	6	5.25	4, 3 and 1.5 in high.
2	11	3	8	3.50	2.5 by 2 in. sq.
3	11	5	13	$2\frac{3}{16}$	$1\frac{5}{8}$ by $1\frac{5}{8}$ in. sq.

The corrugated rolls, previously described in our *Transactions*, have not proved as successful as was expected in increasing the percentage of prepared sizes, and have largely been replaced by the steel teeth, or the segment-roll with pointed teeth. The slow-speed roll is just being tried, and its superiority over other types remains to be definitely decided; but results so far are favorable to it.

2. *Cleaning.*

The methods of cleaning coal may be divided into: (*a*) hand-picking, upon stationary chutes or tables; or upon moving tables; and (*b*) treatment by automatic pickers, depending on the difference in specific gravity; or the difference in coefficient of friction between coal and slate; or the fracture, or shape into which the minerals break.

a. Hand-Picking.—The stationary table or chute is arranged so that the coal gravitates down an inclined plane, with men or boys located at convenient places to pick out the impurities. Fig. 11 shows the plan and sections of a stationary picking-chute at the head of the breaker, which has the advantage of low first-cost and maintenance, and, as is sometimes claimed, increased picking-capacity over other types. All the coal must pass by all the pickers, and should receive a thorough inspection. The disadvantages are: (1) when a large quantity of coal is being fed, the men often allow the coal to rush over the table without a thorough inspection, or hold back the coal, restricting the capacity. It is also claimed that much of the slate is hidden under the depth of coal. This, if true, only happens when the capacity of the chute or table is exceeded, so that the coal is not allowed to spread in one layer. (2) The stationary chutes require a greater height of the breaker than the moving table.

The moving table, like the stationary chute, is arranged for the removal of impurities by pickers stationed along the sides. But, since the table always moves, there can be no delay or holding back, restricting the capacity. It may be arranged to run faster in case of a rush of coal. The best method is to retain the coal in a storage-hopper, back of the table, and feed a uniform supply equal to the table-capacity, which must be designed for the maximum capacity of the breaker.

The advantages are: (1) thorough inspection, since all the coal must pass by all the pickers; (2) the action of the moving table as a feeder to the machinery following; (3) the decrease in required height of breaker, as compared with gravity picking-chutes. The table is usually horizontal, but may be installed on an incline, saving height in the breaker.

The disadvantages are the higher first-cost, and the greater expense of maintenance, compared with that of the stationary table.

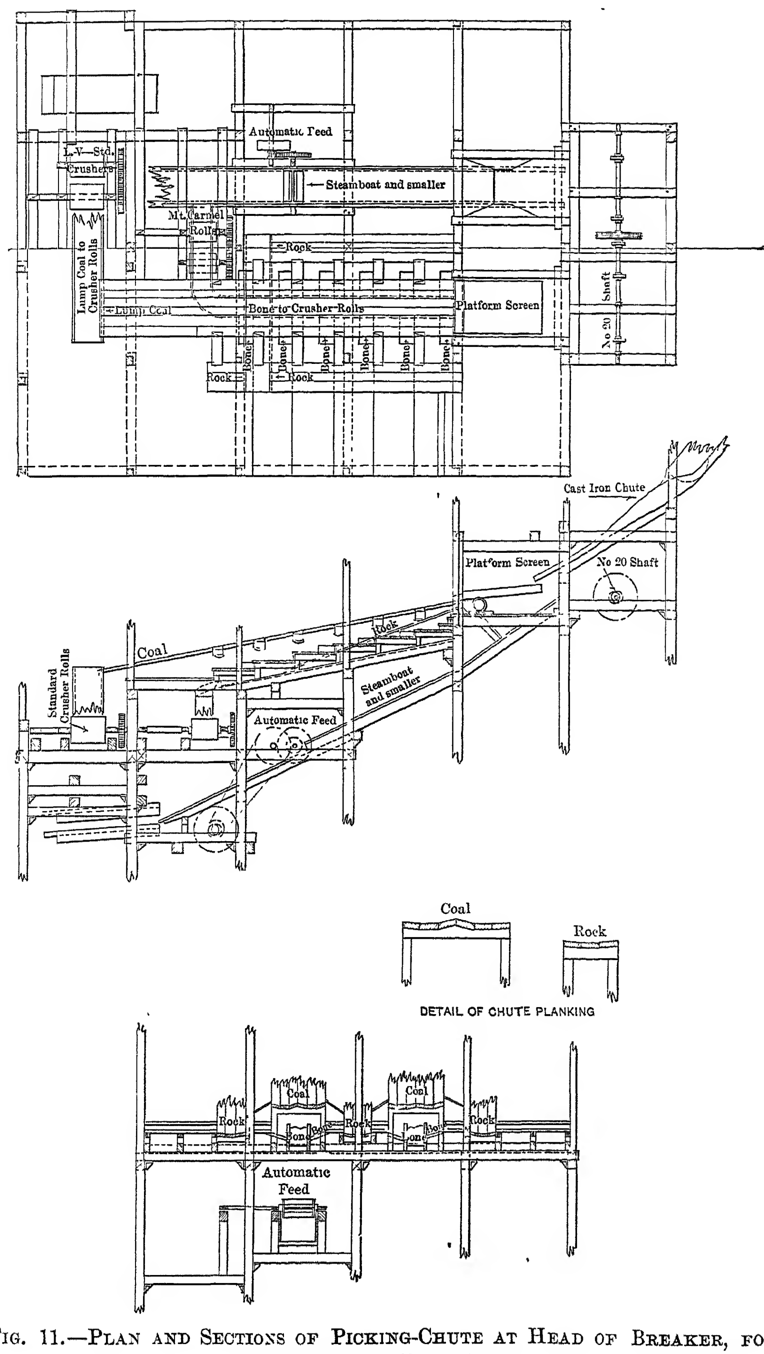

FIG. 11.—PLAN AND SECTIONS OF PICKING-CHUTE AT HEAD OF BREAKER, FOR HAND-PICKING LUMP-COAL.

b. Automatic Pickers.—The most common method of cleaning coal, depending on the relative specific gravity of the minerals in the run-of-mine, is by jigging. The type of jig shown in Fig. 12 may be erected in batteries of any number, the tank being built long enough for the number of jigs required, and divided by partitions at the distances which will give the proper inside dimensions. A cast-iron plate divides each jig-tank into two parts, forming a compartment in the rear, in which the plunger operates. This compartment is preferably lined with phosphor-bronze rubbing-plates, reducing the friction and wear on the adjusting-strips on the edge of the plunger, and maintaining a close fit, preventing slip and the passage of water around the plunger, which would reduce the water-displacement at each stroke and decrease the efficiency of the jig. The front compartment is divided by the regulating-gate. The coal, fed behind the gate, must pass down and under it, and over the perforated grates. The water, forced down by the plunger, rises through the perforated jig-grates, which extend from the bottom of the division-plate to the front of the jig, and raises the mixture of coal and slate off the grates, allowing the heavier minerals (slate and heavy "bone") to settle to the bottom, while the coal rises to the top. The grates pitch 1 in 12 from the division-plate to the front, causing the material to move forward and downward at each stroke of the plunger. The coal is drawn or skimmed off by a steel conveyor-flight, which conveys the coal up a cast-iron inclined trough, allowing the water to drain back, and discharges the coal at the top into inspection-chutes leading to the loading-pockets. The slate and refuse are drawn off in a similar manner by a conveyor located under the coal-line. At the discharge-end is an inspection-chute, from which the slate is conveyed to a refuse-pocket for further disposition.

The plunger is driven by two eccentrics on a shaft directly over the jig-tank. In the earlier forms of jigs, the eccentrics were driven by sliding-links or elliptical gears, arranged to give a quick down-stroke with a slow return. Tests did not show any superiority over the present type of construction, and the former design is now obsolete. The present arrangement has an eccentric within an eccentric, held in place by bolts,

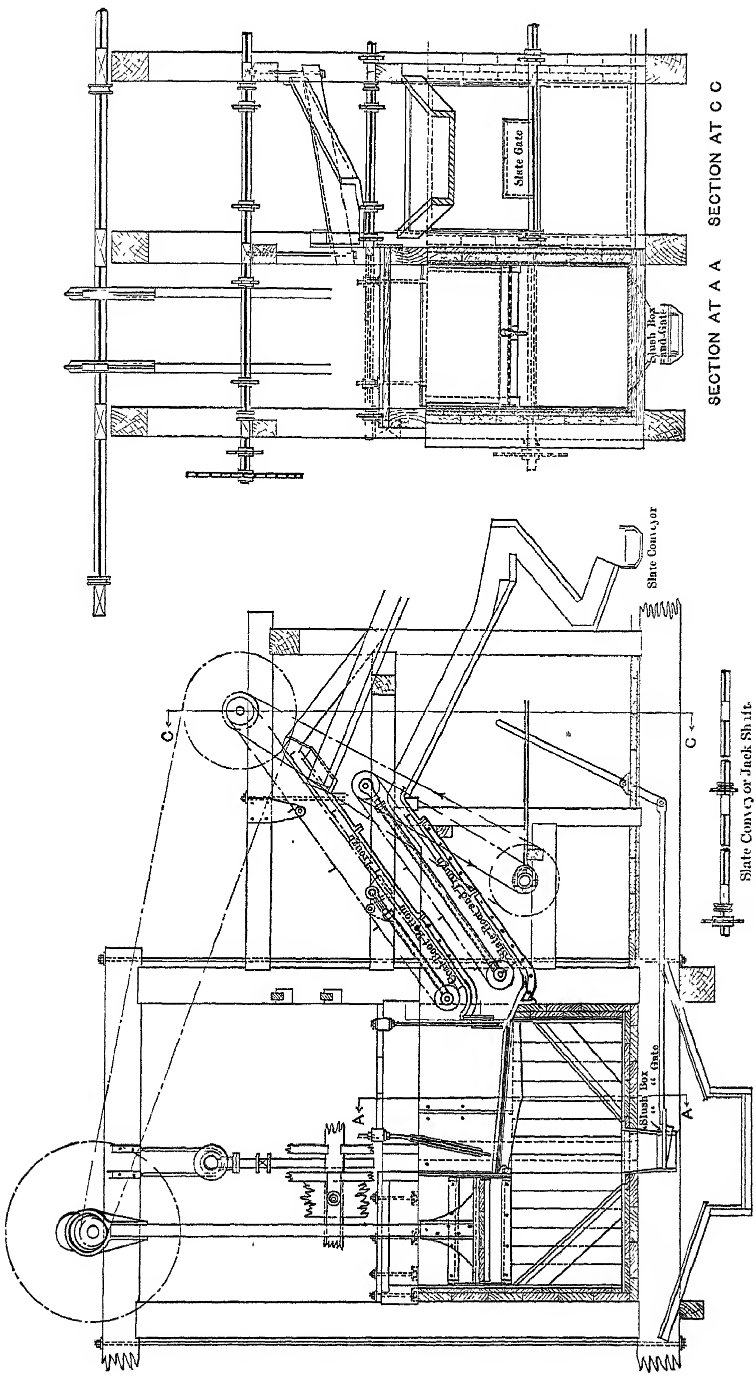

FIG. 12.—SECTION THROUGH CENTER-LINE OF JIG.

and so arranged that the stroke may be adjusted by revolving the outside eccentric on the inner one. This allows a change in travel of from 2 to 6 in.—the shorter throw being used for the smaller sizes (pea and buckwheat), and gradually increasing up to the larger (broken and egg). Tests of the stratified mixture in the jigs seem to show that most of the cleaning or separating of the coal and slate takes place as the mixture passes under the regulating-gate, which may be raised or lowered to accommodate each size of coal jigged. In order to direct the greatest impulse to that point, the area of the openings per square foot of grates is largest at this point and smaller in front of the opening leading to the slate-discharge.

The coal-conveyor head-shaft is driven direct from the eccentric driving-shaft, by means of a chain-drive. A counter-shaft is driven from the coal-conveyor head-shaft. This latter shaft drives the slate-discharge conveyor by a chain, which may be thrown in or out of operation by means of a spiral-jaw clutch, the operating-lever being located within easy reach of the jig-tender. The hutch-product, which falls through the grates, is slushed or washed out when the slush-gate is opened, passes over a jig slush-shaker, and is returned to the sizing-screens.

More or less breakage occurs in a jig, for which the coal-conveyor is largely responsible; and recently I have replaced the steel flight and chain with a belt-conveyor. This has decreased the degradation; but the increase in cost of maintenance may exceed the saving in breakage. The test is not yet concluded. The belt-conveyor has the additional advantage, especially on the larger sizes of coal, that it may be used as a moving table for the inspection and picking of the jigged coal before it is delivered into the pockets.

Table XI. gives a comparison of the saving in breakage by use of a belt, instead of a chain-conveyor, to remove the coal from a jig. The test was conducted on a jig for broken-coal. The first result shown is the breakage made in the jig-tank by the rubbing or moving of the coal on itself, and cannot be charged against the method of discharging the coal.

TABLE XI.—*Percentage of Smaller Sizes Made in Jigging Broken Coal.*

Stove.	Nut.	Pea	Buckwheat	Rice.	Barley.	Dirt	Pea and Smaller.	
0.42	0.21	0.21	0.28	0.16	0.29	0.84	1.78	Breakage made in jig-tank.
1.05	0.64	0.58	0.67	0.37	0.56	1.55	3.73	Steel flight-conveyor.
0.92	0.49	0.75	0.79	0.45	0.73	1.74	4.46	Steel flight-conveyor.
0.85	0.44	0.44	0.55	0.27	0.44	1.07	2.77	Belt-conveyor.
1.16	0.40	0.56	0.62	0.36	0.56	1.50	3.60	Belt-conveyor.

The breakage shows a loss of about 10 cents a ton for jigs equipped with flight-conveyors, and 8 cents a ton for jigs equipped with belt-conveyor, or a saving of 2 cents a ton by using the latter.

Table XII. shows the loss in jigging stove-coal in a jig equipped with steel flight conveyor-discharge.

TABLE XII.—*Percentage of Smaller Sizes Made in Jigging Stove-Coal.*

Pea	Buckwheat.	Rice	Barley.	Dirt	Total.	Loss in Cents Per Ton.
0.4	0.3	0.1	0.2	0.3	1.3	3.25
0.29	0.35	0.14	0.21	0.58	1.57	4.25

Table XIII. shows the capacity of the jig in removing impurities. The tests were made on stove- and chestnut-coal, and the amount handled was from 10 to 12 tons per hour. The average specific gravities were: coal, 1.63; slate, 2.32; bone, 1.92.

TABLE XIII.—*Impurities Before and After Jigging.*

	Before Jigging.			After Jigging.			
	Percentage of			Percentage of			
Size.	Slate	Bone.	Slate and Bone, Half-and-Half.	Slate.	Bone.	Slate and Bone, Half-and-Half.	Coal in Refuse
Nut.....	23.25	3.25		3.50	3.25		4.25
Nut.....	20.5	6.50	0.75	2.25	3 00	0.5	5.00
Stove.	25.5	5.00	0.75	1.5	2.25	0.5	6.00

When the slate or refuse in a jig is drawn out by means of the slate-conveyor, any coal drawn with the refuse may be reclaimed by hand-picking in the refuse-inspection chute. The percentage of coal in the refuse, as shown in the above jig-tests, could be considerably reduced by this method. Any refuse drawn out by the coal-conveyor may be reclaimed in a similar manner—the jig-tenders or pickers removing by hand sufficient of the refuse in the coal so that the product going to the loading-pockets will not contain a greater percentage of impurities than is allowed.

The capacity of a jig for removing impurities depends largely upon the plunger- or water-displacement per minute. In some styles of jigs, where the plunger operates in a box lined with cast-iron rubbing-plates, it is very difficult to maintain a tight fit, to prevent slip, or the flowing of water past the plunger at each down-stroke, which reduces the water-displacement and capacity. To the plunger are fitted adjusting-strips, which are set out against the rubbing-plates, to secure a perfect fit. When cast-iron plates are used, the adjusting-strips should be gone over and set out at least once every day. It has been found that, when bronze rubbing-plates are used, the plunger-strips require adjusting only about once every week, and will maintain a tight fit for that length of time. This advantage in jigs using bronze for rubbing-plates has increased the capacity of that design.

TABLE XIV.—*Capacity of Jig in Tons Per Hour.*

<table>
<tr><th>Rubbing-Plates.</th><th>Broken.</th><th>Egg.</th><th>Stove.</th><th>Nut.</th><th>Pea.</th></tr>
<tr><td>Cast-iron</td><td>............</td><td>4</td><td>4</td><td>5</td><td>6.5</td></tr>
<tr><td>Bronze</td><td>6</td><td>8</td><td colspan="2">10 to 12</td><td>12 to 15</td></tr>
</table>

The capacity of the latter type, as shown in the table, is higher than is usually obtained under ordinary working-conditions, which reduced it about 25 per cent.

There are many machines for cleaning coal which depend on the angle of friction. One of the first types installed (Fig. 13) consisted of a series of slots at right-angles to the center-line of the chute. The mixture of coal and slate, in sliding down the

inclined chute, is accelerated. The coal, with smaller coefficient of friction than slate, gains in velocity, and is carried over the slot or opening by virtue of its greater momentum. The openings may be adjusted for different varieties of coal; but when the coal is from seams of different qualities, or when there is an alternate wet and dry mixture coming from the mines, the slots must be opened wide, in order to pick out sufficient impurities to pass inspection; and in these cases too large a proportion of the coal goes with the refuse, resulting in a reduction of mine-car yield, which is further reduced by the excessive breakage.

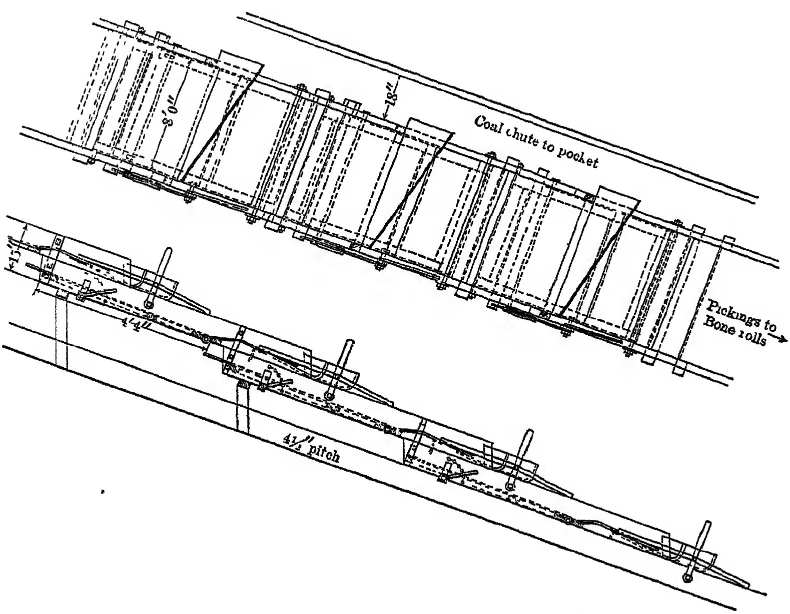

FIG. 13.—PLAN AND SECTION OF GRAVITY PICKER.

The spiral separator consists of a center column, with a series of spiral bands, inclining towards the center, down which the coal and slate slide. The coal maintains a fixed path as long as the friction of the coal on the chute and the centrifugal force balance. As the velocity increases, the centrifugal force increases, and when it overcomes the friction, the coal moves to the outer edge of the spiral plate and falls into a coal-chute. The slate, with higher coefficient of friction, follows a regular path, and at the bottom goes to the refuse-pocket. This type

will work fairly well when handling all dry or all wet coal, but will not give good results on a mixture of both. It is sometimes used on the mud-screen run-of-mine for a preliminary separation, to remove solid rock before jigging. The capacity is about 4.5 tons per hour. The falling of the coal from the spiral plate to the coal-chute causes a loss in breakage estimated at 3 per cent. This type, as well as the former, requires considerable height for installation, and may increase the height of the breaker.

A type of picker recently installed for anthracite consists of a metallic moving picking-table, which is adjustable, to give it a pitch in two directions, first, across the table, and second, inclined on the center-line, so that the moving band travels up the pitch. Coal is fed to the table at the high corner and travels obliquely across the table, discharging over the opposite side, the pitch being sufficient so that the resultant of all forces down the table is greater than the friction of coal on the moving table, tending to carry it up and over the slate-discharge end. The slate, by reason of its greater coefficient of friction, is carried up and over the discharge into the refuse-chute. This type, on larger sizes, requires an attendant, and might be described as a mechanical moving picking-chute. The operator merely touches the slate, increasing its adhesion to the moving band, when it is carried away. This saves the lifting and handling needed on an ordinary picking-chute, and increases the efficiency of a man, or his capacity for inspection and cleaning.

Mechanical pickers depending on the fracture of the coal or slate are installed when the run-of-mine contains a high percentage of flat material, which must be removed from the sized coal to improve its appearance, but would not materially affect the burning qualities of the coal. One type has already been described above under "Fixed Screens." In general, they consist of a series of long and narrow openings, over which the more cubical pieces of coal slide, allowing the flat material to fall through. Such coal as falls through may be cleaned of impurities in a separate jig or other separator; the product is then broken down into one of the smaller sizes and re-screened.

Table XV. gives the average number of pickers of all kinds, including jig-tenders, required to clean 1,000 tons of prepared sizes of coal in 10 hours.

TABLE XV.—*Number of Pickers Per* 1,000 *Tons in* 10 *Hours.*

First Class	Second Class.			Third Class
Dry	Dry and Wet			Wet.
Hand Loading and a Clean Run-of-Mine.	Hand Loading and a Clean Run-of-Mine.	Chute-Loading and Dirty Run-of-Mine. High in		Chute-Loading and a Run-of-Mine High in Slate.
		Bone and Slate.	Slate.	
19.2	23.	66.8	47.	42.

3. *Elevators and Conveyors.*

Conveyors are used to transfer material from one point to another, either in a horizontal or an inclined plane. The inclination should not exceed 30°, and the average is much less. An elevator is used to transfer material vertically, when the point of discharge is vertically over the point of inlet, or nearly so.

Conveyors may be classified according to style of chain, flight, or trough, but I will name only three general classes: (1) single-strand conveyors; (2) double-strand conveyors (comprising scraping- and carrying-conveyors); and (3) belt-conveyors.

The single-strand conveyor is recommended for all sizes of flights up to and including 8 in. wide by 18 in. long. The length is variable and depends upon the strength of the chain and the factor of safety adopted. The double-strand conveyor is recommended when the size of flight required for the capacity exceeds the size to be used for a single-strand conveyor.

The single-strand conveyor will not take an over-capacity without burying the chain in the material, which will often ride the chain, and being carried under the head-sprocket, throw off the chain or break the line, causing delays. By using a single round-bar chain, this disadvantage may be eliminated.

The double-strand conveyor may be constructed so that the chain will be up and out of the trough and be protected from the material when overloaded.

The carrying-conveyor is used, generally, to convey run-of-mine coal from the dump to the head of the breaker. The capacity is limited, and the speed slow, 60 ft. per minute being the maximum.

Belt-conveyors are not extensively used. They have the advantage of large capacity at small horse-power, but must necessarily discharge all the material at the end unless a tripper is used—which requires room not always available. Rubber belts depreciate when idle, especially after having been subjected to the action of sulphur-water; and the long periods of suspension during dull months are hard on rubber belts.

Fig. 14 gives detail cross-sections for 6- by 16-in. and 8- by 24-in. cast-iron flights and cast-iron trough-conveyors, especially desirable where the material to be handled is saturated with acid mine-water, which would quickly attack a steel trough. The trough is cast in sections, 4 ft. long, and bolted together. For a double-strand conveyor, a 2.5- by 2.5- by $\frac{3}{8}$-in.

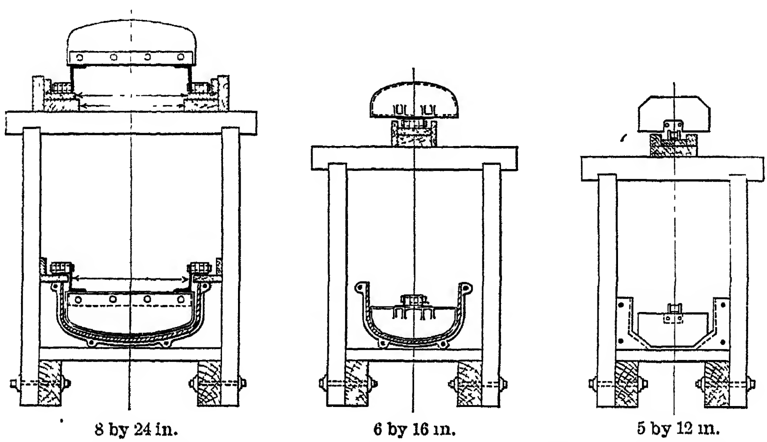

FIG. 14.—CROSS-SECTIONS OF SINGLE- AND DOUBLE-STRAND CONVEYORS.

angle is riveted to the flight, to which is bolted a 5- by 3-in. chain-attachment angle. The lower or conveying chain is attached to the 5- by 3-in. angle in such a manner as to be up and out of the trough, and is carried on a hard-wood strip. This method reduces the wear on the trough, but the wear is not severe on the chain, when the hard-wood strip is lubricated. The flight cannot tilt or bend backward, as is often the case when the chain is unsupported on the lower run. The return-run of the conveyor is carried in a similar manner to the lower run. To handle material under ordinary conditions, I have found that three sizes of conveyors are ample for all the requirements of a modern breaker.

TABLE XVI.—*Sizes of Conveyors.*

Size of Flight	Centers.	Capacity at 125 ft. per Min.	Number of Strands	Average Chain-Pitch	Average Diameter of Sprocket.
Inches.	Inches.	Tons.		Inches.	Inches.
5 by 12	18 to 24	40	Single.	3	30
6 by 16	24	90	Single.	6	30
8 by 24	27 to 36	175	Double.	6 to 9	30

Larger conveyors are often used to handle the run-of-mine from the dump into the breaker; but such installations are special.

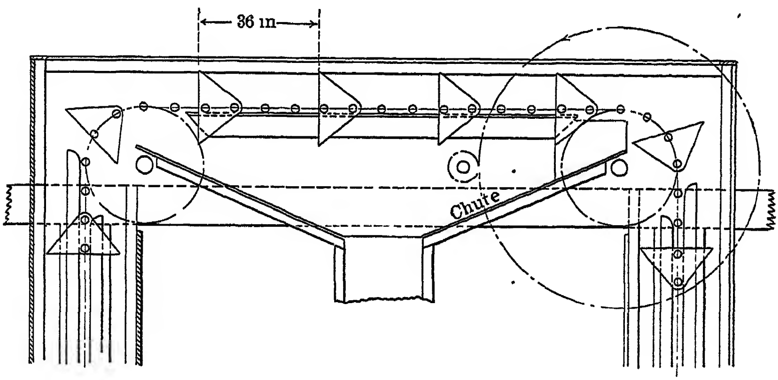

FIG. 15.—SIDE-ELEVATION, SHOWING DOUBLE-STRAND GRAVITY-DISCHARGE ELEVATOR.

Elevators may be of two types, single or double strand. There are many sizes of buckets and styles of chain on the market. The bucket and chain used depend on the capacity and height required. A single-strand elevator should not exceed 50 ft., or a double-strand elevator, 75 ft., centers of sprockets. Fig. 15 shows a 22- by 24-in. bucket, gravity-discharge type. This size will meet all the requirements of a 2,500-ton breaker.

The loss in breakage by handling coal in scraper-conveyors is not excessive, usually not exceeding 2 to 3 per cent., and generally less. Practically all the loss is due to the method of chuting the coal into the conveyor; little or no breakage being due to the conveying or discharging of the coal. With belt-conveyors, the breakage at the discharge-end, due to the drop at high velocity, is excessive. In elevators, the loss in

breakage is high, by reason of the method of filling the buckets at the foot, and the drop at the discharge. In a modern breaker, the design should be such as to eliminate the handling of prepared sizes by either conveyors or elevators, except in the case of condemned coal.

4. *Chutes.*

Coal-chutes are used to convey material from a higher to a lower point by gravity. If badly constructed, they cause a loss by degradation of size, which often exceeds the combined losses from all other sources. The quality of coal varying, the pitch on which it will slide must be changed to suit local conditions. Clean coal will run on less pitch than the mud-screen mixture of coal and slate of the same size; hence, each particular chute must be built to accommodate the coal which it is to handle. The breakage of coal in gravity-chutes may be attributed to the following sources: irregularities in the bottom of the chute; the striking of one piece of coal against another; drops at any point, especially at right-angle turns; and the blow which the coal receives at such turns when it runs against the side of the chute, or when one piece of coal strikes another. An ideal chute should eliminate the above features, thus reducing the breakage and increasing the mine-car yield.

Chutes may be divided into two classes, inclined chutes and vertical telegraphs. The former are used to convey coal from a higher to a lower point not directly beneath the starting point, and the latter are used to lower coal vertically.

A proper chute is generally rectangular in cross-section, and of a pitch which will just allow the coal to start running after it has been stopped or held back. The corners or turns are banked or raised so that pieces will slide around without striking the sides and without drop. When the chute makes a 180° turn, a back-switch is recommended, so that the coal may be brought practically to rest, and the velocity reduced, before starting down the pitch again.

Table XVII. gives the size of coal, the usual pitch in inches per foot, and the size of chute usually used. The lining of the chute is sheet steel for all sizes below broken. The lump, steamboat, and broken will slide on smooth cast-iron plates, inclined on the pitches shown.

TABLE XVII.—*Pitch and Width of Chutes.*

	Pitch	Width
	In per Foot	Inches.
Lump	2.25 to 2.5	36
Steamboat......	2.25 to 2.5	36
Broken...... ..	2.5 to 3	30
Egg	2.625 to 3.25	24
Stove..........	2.75 to 3.5	24
Nut....	3 to 4	18
Pea	4 to 5	12
Buckwheat ...	4.5 to 6	12
Rice	5.5 to 6.5	12
Barley	7 to 8	12
Dirt.	8 and over	12

After steel-lined chutes have been installed in a breaker, it is sometimes found that the coal blocks at certain places, by reason of lack of pitch. When it is not practicable to change. the pitch of the chutes, the steel lining may be replaced with bronze, or sometimes with glass, on which the coal will run at a much lower pitch. A test made in a bronze-lined chute 18 in. wide showed that, for the passage of 2 tons per minute, the following pitches were required: for egg, 2.75 in.; for nut, 3.75 in.; and for barley, 4.2 in.

The vertical telegraph is of two kinds, the first of which is employed simply to lower coal, while the second both lowers coal and stores it in a pocket.

The first type (Fig. 16) consists of a wooden box, open on the sides. Shelves are placed alternately on the front and back inside the box, and inclined towards the center. The spacing between shelves is just sufficient to allow the coal to pass. The shelves are covered with steel, so arranged that the opening between shelves may be opened or closed by adjusting the steel plate. In this type, coal enters at the top and leaves at the bottom.

The second type (Fig. 17) shows the same box-construction; but, since this telegraph is used to fill a storage-pocket, provision is made to discharge at various openings in the sides, as the apex of the pile increases in height, blocking the lower holes. The shelves are placed horizontally, with a high back. The falling coal fills these shelves, and, after they are filled, the coal rolls down on the incline of the coal in the shelves, similar to the avalanching of coal on the face of a high storage-pile. Tests show that when coal rolls down on a pile, the velocity is

uniform and the breakage practically *nil*; and this telegraph was designed upon that principle. The shelves are so spaced that a line tangent to the angle of repose of the coal on any shelf will be tangent to the back of the opposite shelf, so that the coal will roll down the telegraph until the filling of the pocket blocks the bottom opening, when the coal will roll down on the tangent-line and discharge through a side-opening.

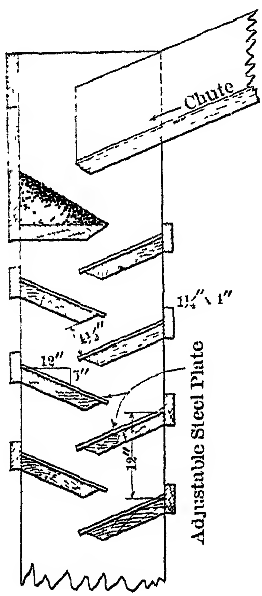

FIG. 16.—VERTICAL TELEGRAPH, USED FOR LOWERING COAL.

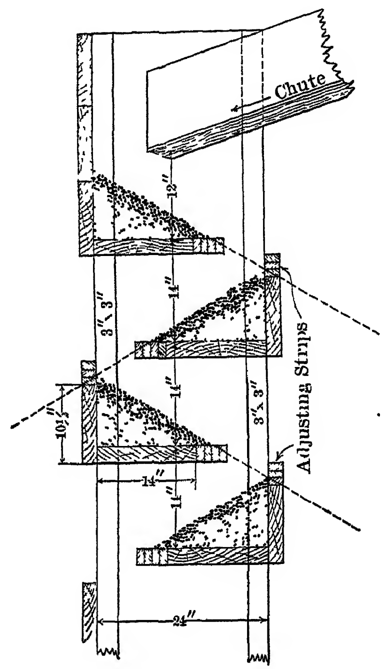

FIG. 17.—VERTICAL TELEGRAPH, USED FOR LOWERING AND STORING COAL.

Tests of this style of telegraph show very little loss—generally under 1 per cent. when properly constructed. They are used for all sizes up to and including egg-coal. They may be of any height. I have designed one of large capacity (300 tons per hour) 65 ft. high. The capacity of a 24-in.-square telegraph is about 150 tons per hour; of one 48 in. square, about 300 tons per hour. The capacity is directly proportional to the width.

5. *Automatic Feeders.*

This class of machinery is used to control the supply of coal during preparation.

The simplest form of feeder used is the hand-controlled gate. Its use is recommended between the dump-chute and the picking-head or room. By using this style, the feed to the men picking may be cut off or held back, so that the men may more thoroughly inspect a certain lot of coal on the table, the quality of which is doubtful; or accelerated when the run-of-mine is good. An automatic feeder delivers a uniform amount constantly, and cannot thus carefully control the supply to the table.

There are several types of automatic feeders. The barrel-feed consists of four plates set at 90°, bolted to spiders rotating on a shaft. This machine is placed in a chute and rotates in the direction in which the coal flows. The coal blocks behind the paddles, and is picked up, carried over, and dumped on the opposite side. The drop on the opposite side in the larger sizes of this machine is high, causing breakage. The feed is intermittent and not continuous, so that the discharge from the feeder comes in rushes.

The reciprocating-feeder consists of a flat table, operated under the discharge-end of a dump-hopper. Its method of feeding is as follows: As the table moves forward, the material on it is carried with it and the material in the hopper crowds down to fill the vacant space. When the feeder moves backward, the material in the hopper prevents the material on the table from returning, and it is discharged over the end. The stroke of the feeder can be adjusted to increase or decrease the feed. This feeder is also intermittent in operation, but delivers the coal with little breakage. It is adapted for feeding the large sizes and run-of-mine coal.

The rake-feed consists of a series of pointed teeth, mounted together, and raised and lowered through the bottom of a chute by means of an eccentric. The teeth, when up, hold back the coal, allowing it to slide over when the teeth are down. It feeds intermittently, and allows the fine sizes of coal to pass by between the teeth, when feeding an unsized mixture.

Of the continuous-feed type, one machine frequently used consists of a round, horizontal revolving table, the coal being

fed through a cylinder, suspended directly over the vertical axis of the table. The distance is such that coal discharging at the end of the cylinder will not roll over the edge of the table. An adjustable scraper located at the edge of the table scrapes the coal into a chute, and as the coal is scraped off, the material in the cylinder rolls out to fill the space made vacant. The feed may be adjusted by moving the scraper in or out, in relation to the center of the table, thus increasing or decreasing the amount fed. There is little breakage made in this type of feeder, and the amount delivered may be easily and quickly adjusted. It is not to be used on run-of-mine coal, but on sizes smaller than broken.

Feeders are very necessary in a breaker, as the control of coal during preparation, especially of that going to sizing-screens, allows the screen-area to be made a minimum, and not necessarily designed to handle a rush of coal, as is so often the case in breakers operating without feeders.

III. Power.

Economical engines to drive breakers have not, until recently, received the consideration they deserve. The demand for small sizes of coal was light, and the overproduction was considered a waste, and used to generate steam at the colliery boiler-plant. The increasing market for steam-coal has made it necessary to economize in the use of steam, by installing high-class and efficient engines. It was formerly not uncommon to find engines using as high as 75 lb. of steam per horse-power per hour. With a modern engine, the steam-consumption should be not more than 23 lb. when running at its full load. The preference is for a compound, non-condensing, Corliss engine. Such a machine should be located in a separate building, free from dust, and arranged to drive all the breaker-machinery, with the exception of the jigs. Jigs, and all the machinery to which they are tributary, should be driven by a separate engine. This arrangement allows the jigs to continue cleaning coal in case the breaker is stopped, the coal being fed from a storage-pocket to the jigs. The main breaker-engine may be belted to a main line-shaft, and power may be distributed from this to the various machines by means of belts or rope-drives. The flexibility of the latter system makes it preferable for long drives and for

isolated machinery. When a break occurs in a rope-drive, the rope can generally be spliced with very little loss of time to the breaker. The splice is known as a "short splice," and is usually from 24 to 30 in. long. Various styles of rope-cones have been devised for quick splicing, but are not to be recommended, as the length will not usually permit them to pass over the smaller sheave-wheel without a bad bend in the rope at the point where the rope enters the cone.

Rope-transmission of power has the disadvantage of increasing the total horse-power required, when many long drives are installed, with frequent turns over idler-sheaves.

A saving in horse-power would result by the use of more efficient machinery in the breaker, by the use of roller-bearing pedestals, etc., as the frictional horse-power required is very close to 75 per cent. of the total power required to drive the breaker when loaded.

Table XVIII. gives the total horse-power required to drive various breakers, and the machinery in operation:

TABLE XVIII.—*Horse-Power Required and Number of Machines Operated at Three Breakers.*

Number	Horse-Power.		Rolls.	Shakers.	Picking-Tables.	Conveyors.	Elevators.	Jigs.	Feeders.
	Light.	Loaded.							
1.	216	280	7	33	2	1—110 ft. 1— 75 ft.	1— 64 ft. 2— 60 ft.		
2.	342	512	4	34	......	1— 75 ft. 1— 60 ft.	3— 50 ft. 1— 60 ft.	32	3
3.	393	546	4	35	.....	1— 90 ft. 1— 75 ft. 1— 60 ft. 1—300 ft	3— 65 ft. 1— 45 ft.	.....	2

The horse-power in the table is the indicated horse-power of the breaker-engine.

In breaker No. 1, the short drives were belted, and the long ones had rope-transmission, installed without the need of any angle sheave-wheels to carry the rope around corners, which would increase the bending-stress in the rope, and the total horse-power of the breaker.

In No. 2, the power was delivered by rope-transmission almost exclusively. Drives to isolated machinery were long,

with many bends or turns in the ropes, and required more power in proportion to the machinery driven than in No. 1.

In No. 3, the method of driving was similar to No. 2. There was a large carrying-conveyor, 300 ft. long, to elevate the coal into the breaker, which required from 60 to 75 horse-power.

TABLE XIX.—*Actual Horse-Power Required to Operate Various Machines.*

Machines	Horse-Power	Speed per Minute.
Revolving-screens, . .	3 to 5	250 ft. periphery.
Shaking-screens, ...	2 to 3	140 to 150 revolutions.
Oscillating-bars, ...	3	50 revolutions.
Rolls, No. 1, 36 in diameter, .	12 to 15	100 to 110 revolutions.
Rolls, No. 2, 36 in. diameter, .	12 to 15	100 to 110 revolutions.

Horizontal Conveyors, 100 ft. Long, at 100 ft. Per Minute.

Size of Flight. Inches.	Capacity per Hour. Tons	Horse-Power
5 by 12	40	3.2
6 by 16	90	5.2
8 by 24	175	9 7

Jigs.

Size. Feet.	Kind of Coal Jigged	Capacity per Hour Tons	Horse-Power.	Revolutions per Minute.
4 by 8	Nut or Stove.	10 to 11	2.21	90 to 95

Elevator.

Size of Bucket Inches	Height Feet	Double Strand	Feet per Minute.	Horse-Power.
22 by 24	80	8 in. pitch, eye-bar chain.	130	13.4

IV. LABOR.

The number of men and boys employed on preparation depends on the capacity of the breaker and the quality of the run-of-mine coal. They may be divided into two classes: (1) those directly responsible for the preparation; and (2) those only indirectly responsible, but necessary for the dumping and loading of the coal and the maintenance of the breaker. Table XX. gives in detail the occupation of the various employees, and the number of tons of all sizes shipped per employee in 10 hours.

TABLE XX.—*Average Number of Tons Per Employee in 10 Hours.*

	Hand-Loading and Clean Run-of-Mine.		Chute-Loading and Dirty Run-of-Mine. High in Bone and Slate	Chute-Loading and Dirty Run-of-Mine. High in Slate.	Hand-Loading and Fair Run-of-Mine.	
	Dry.	Dry and Wet	Dry and Wet.	Dry and Wet.	Wet.	Wet.
Breaker-boss........	1,860	1,938	1,030	1,940	1,120	1,330
Ticket-takers....:..................	1,860	1,610	1,030	....	....	1,330
Dumpers.....................	930	570	628	1,940	560	665
Plate-men and table-tenders.. .. .	206	182	150	255	280	221
Skinners and chippers..........	618	420	262	390	....	....
Roll- and elevator-tenders	618	1,380	788	1,470	1,120	.. ..
Hopper- and conveyor-tenders....	930	1,080	628	840	373	....
Screen- and shaker-tenders.........	1,860	1,201	788	980	1,120	1,330
Screen- and picker-bosses......... .	1,860	880	313	393	560	1,330
Hand slate-pickers....................	144	149	51	66	224	1,330
Mechanical-picker tenders	930	1,201	....	....	1,120	1,330
Jig-tenders.............................	....	405	63	184	140	166
Breaker-oilers	930	969	628	650	560	1,330
Breaker, jig and pump engineers,	1,860	1,938	788	583	1,120	665
Loaders....................	156	140	150	215	224	333
Court-house man.........	1,860	1,938	....	1,940	....	1,330
Sweepers and cleaners	930	4,800	1,030	970	560	1,330
Coal-pickers in refuse...............	....	..... .	930	....	....	....

The figures in the last column are for a breaker recently erected and put in operation; but, up to the present time, the estimated maximum tonnage of 1,200 tons in 9 hr. has not been shipped. It will not be necessary to increase the number of employees to clean and prepare the coal when operating at full capacity; and the number of tons per employee is based on the present force and an estimated tonnage.

It will be noted that in this column the number of tons per loader in 10 hr. is much greater—in some instances more than double the number given in other columns. This is the result of a new method of transferring the coal from the storage-pockets into the railroad-cars, which is operating successfully at one of the anthracite storage-plants to load coal into cars, and is one of the features of the new Mineral Spring breaker of the Lehigh Valley Coal Co., near Wilkes-Barre, Pa.

The problems of loading at a storage-plant and at a mine differ. At the former, shipments may be arranged so that all

orders for one size of coal can be loaded in sequence. At the latter, arrangements must be made to load all sizes as they are prepared in the breaker.

Breaker-pockets should hold at least the capacity of the largest car furnished by the railroad to be loaded. Under the old method, with one loading-track, and pockets located parallel to the center-line of track, simultaneous loading from adjacent pockets cannot be done, unless the distance from center to center of loading-gates is equal to or greater than the over-all length of a car. This is not usually the case; and so it often happens that during loading from one pocket, an adjoining bin becomes filled with coal, blocking the chutes leading to the pockets and causing a delay throughout the entire plant. Such a condition may be remedied either by installing one or two additional loading-tracks, or by increasing the pocket-capacity.

Fig. 5 shows a section through the loading-pockets of a typical modern breaker. There are three tracks, for loading box-cars, low and high gondola-cars, respectively. An additional advantage of separate tracks is that the loading-chutes are arranged to suit each general type or size of car. This arrangement reduces the breakage resulting from a high drop of coal when loading into a low-side car from a pocket arranged for loading a high one. A hinged loading-apron or chute may be used, which can be lowered into a car; but this effects little, if any, economy. Since the upper end of the apron is fixed, the lowering of the other end into a car increases the pitch, allowing the coal to exceed its normal velocity when sliding over the apron; and the anticipated saving in breakage is usually lost. In the new Mineral Spring breaker, there is a radical departure from former designs in the loading-arrangement. The object desired is to reduce the number of car-loaders required to load an equal tonnage by the former methods, and by careful handling of the material during the preparation, to eliminate lip-screens and the consequent necessity of a final washing of the coal at the point where it enters the car. Washing coal on the lip-screens is usually required to remove the fine screenings, made during preparation, which adhere to the coal. This additional water is very objectionable. It saturates the coal loaded into a car, and freezes during the cold weather. The

water dripping from the loaded car washes the track and road-bed, and helps to make the vicinity of the loading-pockets a very difficult place to keep clean, especially during freezing weather.

The 14 loading-pockets at Mineral Spring are located at a right angle to the center-line of the loading-tracks, and symmetrically about the center-line of the breaker, seven on each side, and arranged to receive seven different sizes of coal, two pockets to each size (egg to barley, inclusive). The capacity of two pockets for egg-coal is about 130 tons, decreasing slightly for each size, down to barley. The loading is concentrated at one point, and large pockets are necessary to store each size of coal coming from the breaker while another size is being loaded. In loading, the coal is fed on to a 36-in. belt-conveyor, located between the two lines of pockets on the center-line of the breaker, and is conveyed to a point beyond the ends of the pockets and discharged into a car, over a loading-apron. The head or discharge-end of the conveyor, together with the apron, may be raised or lowered to suit different heights and styles of cars, by means of a steam-cylinder, controlled by a four-way valve. Another steam-cylinder, controlled by a throttle-valve, opens and closes the pocket-gates through a system of levers and shafting.

Pockets containing the same size of coal are opposite each other, with four feeding-gates, two to each pocket. These four gates are opened simultaneously, and the opposite gates feed into a bifurcated chute, discharging the coal on the belt in the direction in which it is moving, and at about the same velocity.

The cars to be loaded are placed on an Ottumwa steam-actuated gravity box-car loader, consisting of a platform, to which cars are clamped, and then tilted by rotation about a fixed center until the platform is inclined at an angle of about 35°. The center of rotation is located at about the point where the apron will enter an average box-car door. The coal falls into the car and flows by gravity to fill the lower end of the car. When that end is filled, the loader is rotated in an opposite direction, and the other end loaded. Gondola-cars may be loaded in the same way. The levers and valves controlling the starting and stopping of the conveyor, raising and lowering its head or discharge-end, and operating the loading-gates and

the box-car loader, are placed so as to be under the control of one man, located at the discharge-end of the conveyor. In addition to this man, there are three car-runners employed in bringing the empty cars to be loaded, and running the loaded cars over the scales, and to the stand-track.

V. Water Used.

Water is used in breakers of Classes II. and III., to clean and wash coal during sizing; on the lip-screens, to remove fine screenings; and in jigs.

Water is not usually directed on shakers handling mud-screen lump-, steamboat-, or broken-coal, which is to be hand-picked or re-broken into smaller sizes, unless the small material in the mixture coming to the mud-screens adheres to the larger pieces, coating them, so that a picker is unable to pass upon the quality of the product he is inspecting.

The number of gallons of water required per minute is equal to the number of tons shipped per day of 10 hr. Thus, a 1,000-ton breaker will require 1,000 gal. of water per minute. It should be pumped into a storage-reservoir at the top of the breaker, and thence conveyed by supply-pipes to the places where it is needed. The reservoir should be large enough to supply the breaker for at least 20 or 30 min. and thus cover ordinary interruptions in pumping. The size of pipe used for a shaker 4 ft. 5 in. wide and 15 ft. long is usually 2.5 in. inside diameter, with two 1.5- or 2-in. branches delivering on the shaker.

Where the water is acidulated, cast-iron pipe should be used.

VI. Costs.

1. *Cost of Breaker.*

The total cost of a breaker-structure varies with the location, the style of construction, and the class or method of preparation. Table XXI., based on the desired tonnage, and covering all labor and material for foundations, breaker-structure, and machinery, represents present conditions.

TABLE XXI.—*Total Cost of Modern Breaker.*

s .	Tons Per 10 Hours.	Construction	Cost Per Ton.	Cost Per Square Foot.
2	2,500	Timber.	$90.40	$9.80
2	2,000	Timber.	88.50	7.20
3	1,200	Steel and wood.	118.00	14.20

2. *Cost of Operation.*

Table XXII.—*Cost of Preparation and Maintenance.*

(All costs are given in cents per ton of all coal prepared and shipped.)

A.—Hand-Loading and Fairly Clean Run-of-Mine.

Class I —Dry.

Preparation.	Maintenance		Total.
Labor	Labor	Material	
6.78	1.41	1.75	9.94

Class II.—Dry and Wet.

8.73	2.40	3.08	14.21

B.—Chute-Loading and Dirty Run-of-Mine.

High in slate and bone. Specific Gravity: Coal, 1.63; slate, 2.32; bone, 1 92.

Class II.—Dry and Wet.

8.46	2.97	4.08	15.51

High in slate, no bone.

7.58	2.53	3.92	14.03

Class III.—Wet.

Specific Gravity: Coal, 1.52; slate, 2.23; no bone

5.9	1.97	3.15	11.02*

The Storage of Anthracite Coal.

BY R. V. NORRIS, WILKES-BARRE, PA.

(Wilkes-Barre Meeting, June, 1911.)

I. INTRODUCTION.

THE anthracite coal trade, with a shipment averaging about 70,000,000 tons per year, differs essentially from other coal business, in the fact that the larger sizes, comprising about 65 per cent. of the total, are used almost exclusively for domestic purposes, principally during the winter months; and that for proper combustion close-sizing is imperative, so that eight sizes are made; broken, egg, stove, and nut, known as prepared sizes, and pea, buckwheat, rice, and barley, known as steam sizes.

These sizes are made in the regular course of preparation, and but little variation in the natural percentage of each is practicable. Thus it is necessary either to work the collieries intermittently, or to dispose of all the varying sizes in their fixed proportions. Unfortunately, the market does not at all times absorb the coal in proper proportions, and rather than interfere with the regular operation of the mines it has been found economical to provide storage for the excess.

While the cellars of the consumers are the great storage-plants of the country, this capacity is not under the control of the trade, except so far as reduced prices during the spring and summer months may tempt the use of this reserve. The same remark applies but with less force to the retail yards, which, though usually small, store in the aggregate a large amount of anthracite.

While every effort is made to utilize to the utmost the individual storage-capacity of the country, there still remains the necessity for taking care of the irregularities of the market by the construction of storage-plants under the direct control of the producers. Such plants are usually the property of the railroads, and are situated at points most convenient from a traffic stand-point.

II. Location of Plant.

Three general types of location are adopted:

1. *Seaboard.*—Comprising plants situated at or near tidewater. This location has the advantage of placing the stock in a place readily available to the consumer, and providing for its distribution with a minimum danger of interference from derangement of transportation or labor-difficulties, and the disadvantage that the capital locked up in the stock is increased by the freight-charges on the coal.

2. *Local.*—Comprising plants situated in or near the anthracite region, usually at points of convergence of traffic from the various operations controlled by one transportation interest. These plants have the advantage of a short haul from the mines, low freight-charges on the coal stored, and the capabilities of shipping to any market, with the disadvantage of possible unavailability during labor-troubles or interrupted transportation, when the need for the stored coal may be exceptionally great.

3. *Interior.*—Comprising principally storage-plants situated on or near the Great Lakes, especially at Buffalo and at points in Minnesota, Wisconsin, and Illinois. Such plants are principally useful in furnishing fuel to the West during the period of closed navigation on the lakes. Included in this class are a few railroad-plants at important junction-points west of the Alleghanies.

The various transportation interests differ largely in the extent to which they have installed storage-plants, the variations being chiefly due to the distribution and character of the trade enjoyed by each. The percentage of total annual output for which storage has been provided varies from more than 15 per cent. for roads with a large Western trade to a little less than 2 per cent. for coal going largely to Eastern markets.

The total capacity of the storage-plants now in use aggregates a little more than 5,500,000 tons, or about 8 per cent. of the annual output, not a month's supply, showing the error of the popular belief that these plants are installed to provide for labor-troubles at the mines.

Before taking up in detail the various types of plants, an analysis of the principles of storage is essential to a full understanding of the advantages and limitations of each type.

III. Factors of Storage.

Prices.—From Saward's *Coal Trade*, the prices for anthracite at New York, for December, 1909, were: Broken, $4.20 to $4.75; egg, $4.95 to $5; stove, $4.95 to $5; chestnut, $4.95 to $5; pea, $3 to $3.25; buckwheat, $2.35 to $2.50; rice, $1.75 to $2; barley, $1.35 to $1.50. These vary but little from the present prices except for the usual reduction of 50 cents per ton on prepared sizes, usually made April 1, and restored at 10 cents per month till the full list-price is again reached, about September 1.

The above prices, with a difference of $1.75 per ton between the prepared prices and the price for pea-coal, indicate very forcibly the large expense involved in breakage from the prepared to the smaller sizes; and further, the market-standards of preparation permit only small percentages of undersize in any size of coal.

While breakage is inevitable in all handling of a brittle substance like anthracite, the causes of excessive breakage are well known.

Dropping Coal.—The breakage from dropping, of course, varies somewhat with different classes of coal, but from extensive tests, made some years ago, the following losses appear to be nearly an average. $D =$ drop in feet.

Size.	Amount of Breakage Into Smaller Prepared Sizes.	Amount of Breakage Into Small Sizes. Percentage.	Amount of Total Breakage. Percentage.
Broken......	3 per cent. + 43/100 D	2 per cent. + 17/100 D	5 per cent. + 6/10 D
Egg...........	4 per cent. + 43/100 D	2 per cent. + 17/100 D	6 per cent. + 6/10 D
Stove.........	2 per cent. + 33/100 D	2 per cent. + 27/100 D	4 per cent. + 6/10 D
Nut..........	....	4 per cent. + 4/10 D	4 per cent + 4/10 D
Pea............	....	2 per cent. + 5/10 D	2 per cent + 5/10 D
Buckwheat,	....	1 per cent. + 25/100 D	1 per cent. + 25/100 D

These tests were made both by dropping carefully-sized coal through measured distances, and by dropping car-loads into pockets in the regular course of transfer at tide-water. While it is impossible to avoid some drop, a great deal of breakage can be avoided by sliding the coal, either by chutes, as is done in the breakers, or on itself, as it has been found that sized coal delivered on a pile adjusts itself by avalanching in large

masses, with but little breakage, rather than by individual lumps rolling from top to bottom with the resulting attrition.

Drawing coal from the bottom of a pile under pressure also results in very heavy breakage. While no figures are available, attempts to draw from under the centers of high piles have resulted in such disastrous breakage as to stop the practice immediately.

Handling prepared coal by scraping-conveyors results in a loss by breakage and attrition into small sizes, varying from 2 to 4 per cent., depending largely on the methods of feed and discharge. From observation, I do not believe that there is appreciable breakage during the transit, as the length of conveyor seems to have no measurable influence on the breakage.

Belt- and carrying-conveyors have certainly no breakage in transit chargeable to them, but the drop at discharge-points is frequently considerable.

Bucket-elevators cause a breakage of prepared coal into small sizes, varying from 2 to 5 per cent., almost entirely from the feed and discharge.

Freezing.—The coal stored in the open air is in winter covered with snow, resulting in surface-freezing, and not infrequently the snow-covered or frozen surface is buried under additional coal, a fair non-conductor, so that even in summer, frozen coal is found occasionally in the interior of the piles. The reloading of frozen coal is always difficult and costly; usually, gangs of men pick the frozen coal loose, at large cost both in labor and breakage. The most efficient method yet devised for handling coal under such conditions is the use of water, preferably hot water; unfortunately, but few of the plants are arranged to take care of the large drainage, which carries much fine coal dirt resulting from such an operation.

Stocking.—The stocking of coal involves the handling of large quantities rapidly and economically, considerable railroad-yard for the handling of cars, and provision for storing the different sizes separately, as well as scales for weighing in the stored coal, unless such scales are provided at the individual collieries.

Reloading.—This involves a plant capable of handling promptly and economically any one of the several sizes which may be in storage; and, further, owing to the necessary breakage in

handling, the rescreening of all reloaded coal, so that the storage-coal is put on the cars in as good condition as freshly-mined coal; a neglect of this results in a market discrimination against storage-coal, which may involve serious allowances in price. The reloaded coal must also be weighed, and ample railroad-yard provided for handling the tonnage on the outbound end of the plant.

Operation.—It must also be remembered that a storage-plant operates most irregularly; sometimes rushed night and day either stocking or reloading, and at other times idle; full or empty sometimes for months at a time, so that low cost of operation when in active use may easily be counterbalanced by high fixed charges, either as interest on the investment, depreciation, or high fixed labor-cost from a permanent force.

IV. Requirements of an Ideal Plant.

An ideal storage-plant should comprise the following conditions:

Storage of each size separately in varying quantities.

Rapid handling into or from storage of any size.

Minimum breakage in stocking.

Minimum breakage in reloading.

Rescreening all coal before shipment from storage.

Preparation of screenings into sizes, and return of these to their proper piles.

Minimum cost in operating-expenses.

Arrangements for handling frozen coal.

Ample railroad classification-yards for traffic into and out of the plant.

Ample trackage through plant, with preferably gravity handling.

Convenient location of plant and facilities for enlargement.

Minimum danger from fire.

Low first-cost per ton of capacity.

No plants thus far constructed comprise all of the above-named features, and the design of any plant is necessarily far from ideal, involving as it does the balancing of the advantages and disadvantages of various types with a constant view to ultimate economy.

V. General Classification of Plants.

Storage-plants vary much in detail of design, but may be generally divided into two classes—non-mechanical and mechanical storage—with the following types:

Non-Mechanical:

(*a*) Level.	Stocking on the surface.	Reloading by hand or steam-shovel.
(*b*) Level.	Stocking from trestles.	Reloading by hand or steam-shovel.
(*c*) Level.	Stocking from trestles	Reloading by tunnel with or without dock-scrapers.
(*d*) Level	Stocking in bins.	Reloading by tunnels.
(*e*) Level.	Stocking by cable-railway and dump-cars.	Reloading by hand or from bins.
(*f*) Hillside.	Stocking from trestles.	Reloading by hand, scrapers, or hydraulicking.

Mechanical:

(*g*) Hillside.	Stocking by traveling-cantilever trimmer.	Reloading by hydraulicking.
(*h*) Level.	Traveling or fixed tramways.	Stocking and reloading by traveling buckets.
(*i*) Level.	Dodge system. Stocking by truss-trimmers in conical piles.	Reloading by swinging conveyors.
(*j*) Level.	Stocking by traveling trimmer.	Reloading by tunnel and traversing-conveyors.
(*k*) Level.	Covered plants. Stocking by fixed trimmers.	Reloading by traversing-conveyors or by tunnel and dock-scrapers.

1. *Non-Mechanical Storage-Plants.*

(*a*) *Dump-Storage.*—The simplest method of stocking large volumes of coal consists in forming a dump on a fairly-level surface, laying temporary tracks on the accumulating stock, and raising and shifting these as the storage grows in extent and height. Reloading is accomplished either by steam-shovels or grab-bucket cranes, operated from the edges of the pile from tracks which are shifted as reloading progresses.

This plan, which fails to fill the first seven requirements of an ideal plant, is only suitable for temporary storage of steam sizes. Only one size can be stored, the breakage is excessive in any event, and prohibitory with prepared sizes, no rescreening is possible, and the cost of operation, not including waste, approximates from 20 to 25 cents per ton handled.

(*b*) *Trestle-Storage.*—A method of storage, Fig. 1, now in general use in retail yards, and also attempted on a larger scale, comprises the construction of a trestle of the height of the proposed top of the pile, over which the loaded cars are dumped, forming a long pile of usually only moderate height, sizes being separated by partitions. Reloading is accomplished usually by hand.

Such storage violates almost every principle of the art, is small in capacity for the cost, expensive in operation, high in breakage from the necessarily considerable drop from the trestle, rescreening can only be done by hand, and is generally costly and inefficient; it does, however, permit the storage of various sizes. Its use should be confined to small retail yards,

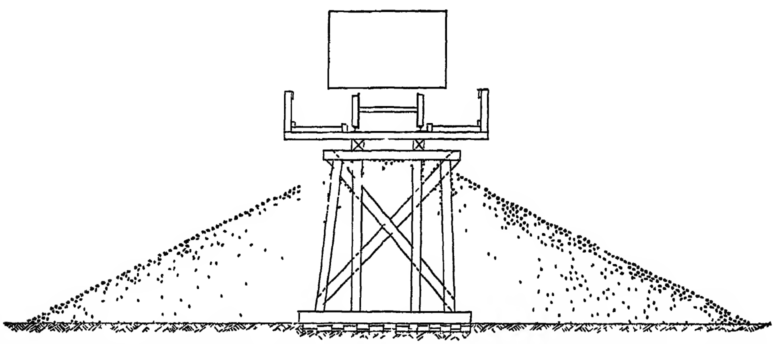

FIG. 1.—STORAGE FROM TRESTLE, WITH OR WITHOUT PARTITION INTO BINS. RELOADING USUALLY BY HAND.

and then is only apparently warranted by a lack of the capital required to install better facilities.

(*c*) *Trestle-and-Tunnel Storage.*—A more efficient type of trestle-storage unites with the trestle-stocking the provision of a tunnel under the trestle for reloading, Fig. 2. The coal is fed into cars in this tunnel through gates, and the cars may be either regular railroad equipment or narrow-gauge dump-cars used for transport to proper screens for final reloading.

Storage-plants of this type comply with the first and partly with the second requirement of an ideal plant. The breakage is excessive, including not only that incident to the trestle-storage, but to drawing at least a portion of the coal from the center of the pile under pressure. Except with the use of separate screening-plant, no rescreening is possible; and further,

less than 60 per cent. of the coal is tributary to the tunnel by gravity, and the two outlying wedge-shaped piles must be transported to the tunnel by hand, or better, by the use of dock-scrapers, which are also occasionally used for extending the storage beyond the gravity-range of the trestle.

A modification of this type is made by installing a conveyor of the belt or scraper type (preferably the former) in the tunnel, which, while it does not reduce the breakage, does reduce the necessary size of the tunnel, and correspondingly the first-cost of the plant.

(*d*) *Bin-Stocking* (Fig. 3).—This is the earliest type of successful storage-plant worthy of the name, and several extensive

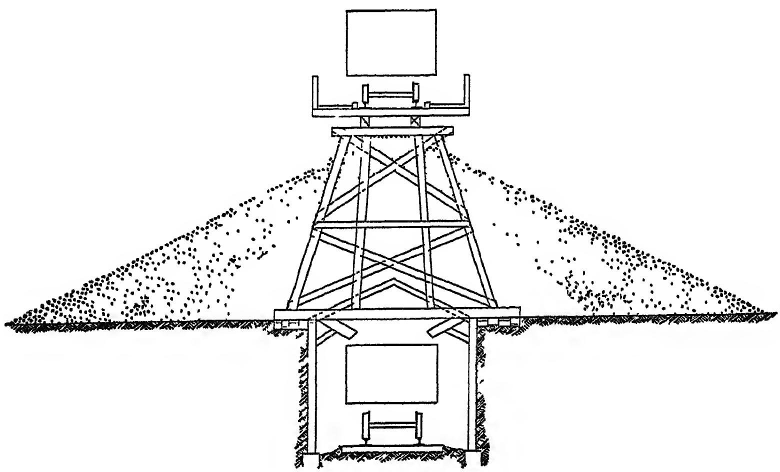

FIG. 2.—STORAGE FROM TRESTLE, WITH OR WITHOUT PARTITION INTO BINS. RELOADING BY GRAVITY AND BY HAND INTO CARS OR CONVEYORS IN TUNNEL, WHICH MAY BE ENTIRELY UNDERGROUND OR BUILT AS A PART OF THE TRESTLE.

plants of this type are still in active service. In general, the construction consists of wooden bins traversed by railroad-tracks, from which the various sizes and types of coal are dumped, each in its appropriate bin. Reloading is usually accomplished by cars passing under the bins, either on the surface or more frequently in tunnels.

To reduce the danger from fire, the movement of the reloading-cars is usually by gravity or by rope-haulage. The individual bins are necessarily limited in capacity to from 50 to 100

tons each, and an extensive plant covers a very large area. One such plant at the seaboard has 384 bins, reloading into cars in nine tunnels, and covers approximately 9 acres. Such a plant costs in excess of $3 per ton of capacity to erect, requires an enormous amount of timber, with resulting large fire-hazard and high maintenance-charges, and the operating-expense approaches 10 cents per ton.

A great advantage is the practicability of storing many sizes and kinds of coal, and keeping separate many small consignments.

This type also includes the majority of the transfer-piers both at the seaboard and on the Great Lakes, where bins of considerable capacity and large in number are used as temporary

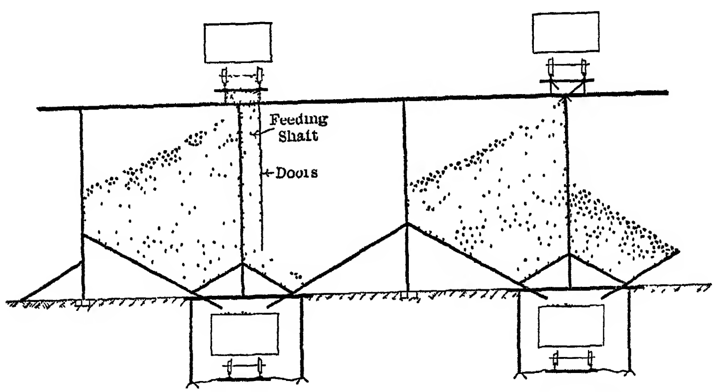

FIG. 3.—BIN-AND-TUNNEL TYPE OF STORAGE. STOCKING FROM RAILROAD-CARS ON TOP OF BINS. RELOADING BY GRAVITY INTO CARS OR CONVEYORS IN TUNNELS.

storage to admit of the rapid loading of vessels without the delays incident to transfer direct from the cars.

The breakage in this type of plant is very serious, caused not only by the necessary drop into the pockets, Fig. 4, but by the drawing under pressure into cars for reshipment; rescreening is impracticable at the plant itself, except on shipping-piers, where imperfect stationary screens are usually installed, and can only be done elsewhere, involving further handling and breakage. Some tests as to the losses involved are available, but these were made with but a few cars in each case, so that the pockets were in no instance filled, and the loss in breakage from the drop does not represent average conditions.

Loss in Undersize in Passing Through Storage-Bins.

Size of Coal.	Breakage Into Smaller Prepared Sizes	Breakage Into Small Sizes.	Total Breakage.
	Per Cent.	Per Cent.	Per Cent.
Broken	19.57	6.60	26.17
Egg	10.18	8.50	18.68
Stove	4.92	8.14	13.06
Nut	....	7.65	7.65
Pea	....	10.83	10.83
Buckwheat	....	4.06	4.06

Even taking half the above-named figures, which would be most conservative, and assuming perfect rescreening, the loss at seaboard on 1,000,000 tons of prepared and pea-coal in about the usual proportions, would amount to $545,000, or

Size.	Original.			Final.		
	Quantity.	Price Per Ton.	Total Value.	Quantity.	Price Per Ton.	Total Value.
	Tons			Tons.		
Broken	130,000	$4.75	$617,000	96,040	$4.75	$456,190.00
Egg	225,000	5.00	1,125,000	191,770	5.00	958,850.00
Stove	195,000	5.00	975,000	192,410	5.00	962,050.00
Nut	200,000	5.00	1,000,000	215,040	5.00	1,075,200.00
Pea	250,000	3.25	812,500	248,630	3.25	808,047.50
Buckwheat	....	2.50	....	35,515	2.50	88,787.50
Rice } Barley }	....	1.75	....	20,595	1.75	36,041.25
	1,000,000	....	$4,530,000	1,000,000	....	$3,985,166.25

The loss in breakage, from this calculation, is 54.5 cents per ton, in addition to the cost of storage.

Of course, in practice, perfect rescreening after storage is not practicable, but a very large amount of rescreening is necessary, involving great losses, which usually fall on the transportation companies, and form one of the items included in their freight-rates.

Many attempts have been made to reduce the breakage involved in handling through pockets, and this is often minimized by the use of shallow pockets, with resultant loss of storage; counter-chutes, spirals, and shelf-chutes in the deeper pockets, and the use of feeding-shafts, shown in Fig. 3, which, when properly maintained and intelligently used, keeping them

full, feeding in at the top as the coal is discharged from the bottom, certainly greatly reduce the losses by dropping.

While rescreening is generally more or less thoroughly attended to separately in this type of storage-plant, re-preparation of the screenings is rarely attempted; the unsized screenings are usually sold under the name of "pea and dust," at a price approximating that of buckwheat-coal.

(*e*) *Cable-Railroad Storage.*—A modification of the bin-and-tunnel type involves the use of cable- or gravity-return cars, running out on trestles over bins or surface-storage, and dumping their contents at the desired points. This type is used at many retail yards and at transfer-points, especially where water-borne coal is transferred to yards or cars. The plant is moderate in first-cost, economical in operation, but high in breakage; does not permit rescreening except as a separate operation, and being of timber is subject to destructive fires. It does, however, lend itself readily to covering for weather protection.

(*f*) *Hillside-Storage.*—At first thought one is inclined to agree with the remark credited to a former president of the Philadelphia & Reading railroad, that there was "no need for mechanical storage in a country full of hills just made to store coal on." Unfortunately, while the anthracite country is full of hills, but very few are even remotely suited for storage-purposes.

The features desirable in a hillside-storage are:

1. A side-slope at least 300 ft. wide by 1,000 ft. long, or more, with a fairly-true surface, and having a pitch between 25° and 30°.

2. That the foot shall change abruptly from this slope to a level surface for the tracks. Most hills end in a vertical curve, changing very gradually from the hillside-pitch to level ground, and involving serious earth-work for tracks.

3. That the surrounding country shall admit of a track to the top of the storage-hill and down again, with reasonable grades, and at moderate expense.

4. Space at the bottom of the hill for reshipping-tracks and yards.

5. A location reasonably suitable for a storage-plant.

I have tramped many weary days looking for just such hills,

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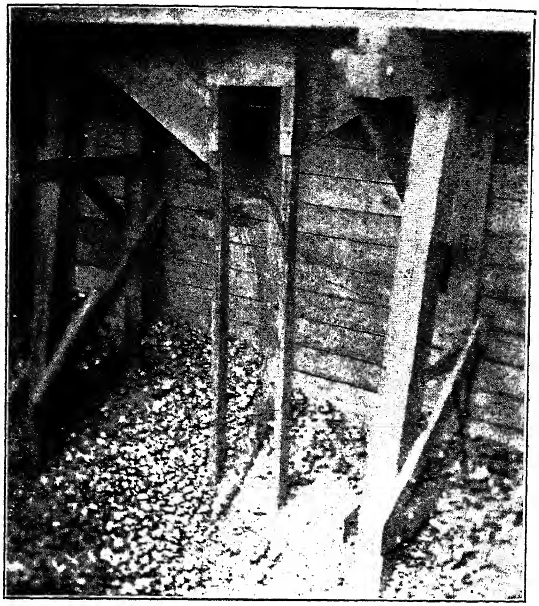

Fig. 4.—Egg-Coal in Storage-Bins, Showing Breakage from Dropping.

Fig. 5.—Hillside Storage-Plant. Reloading. Showing Partition, and Hydraulic Handling to End of Conveyor which Delivers the Coal into a Cross-Conveyor to Screen-House.

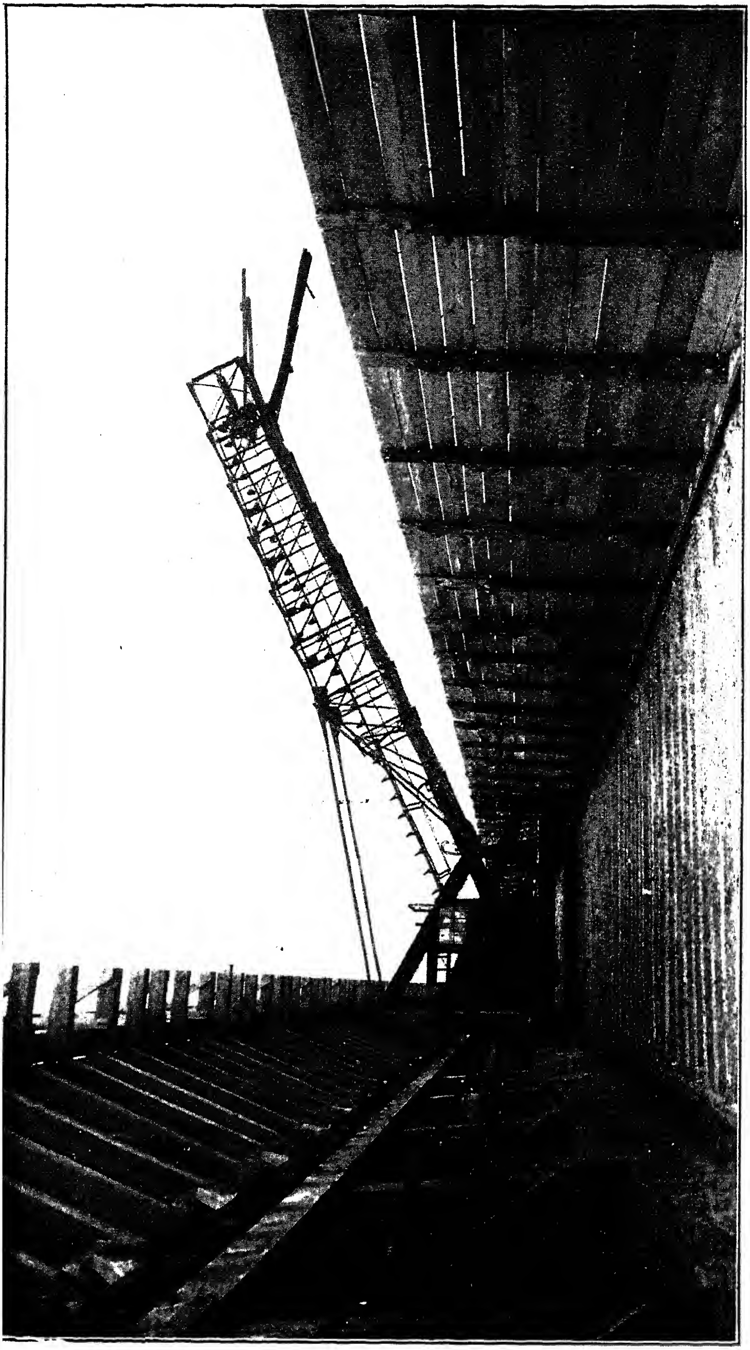

FIG. 7.—TRAVELING CANTILEVER-TRIMMER, HUDSONDALE STORAGE-PLANT. SHOWING FACE OF TRESTLE AND BULKHEAD.

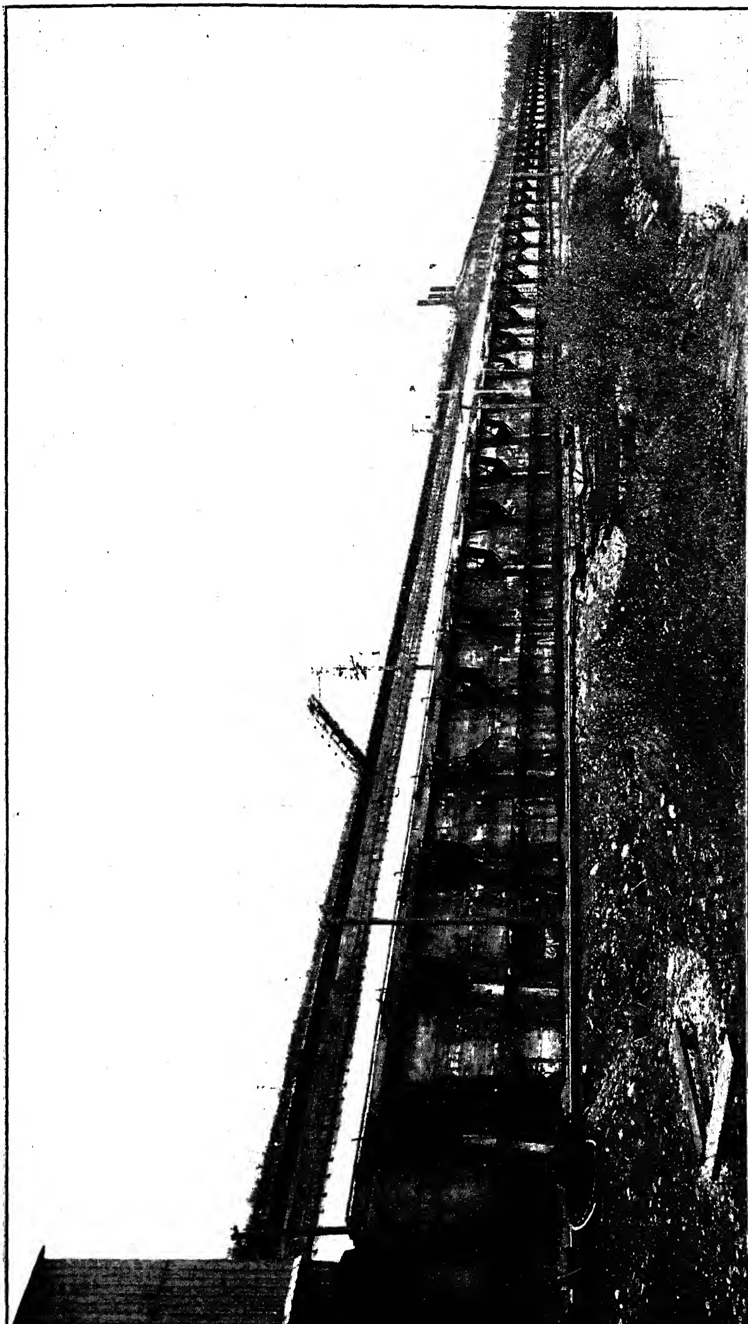

FIG. 8.—CONCRETE RETAINING-WALL WITH LOADING-CHUTES. HUDSONDALE STORAGE-PLANT.

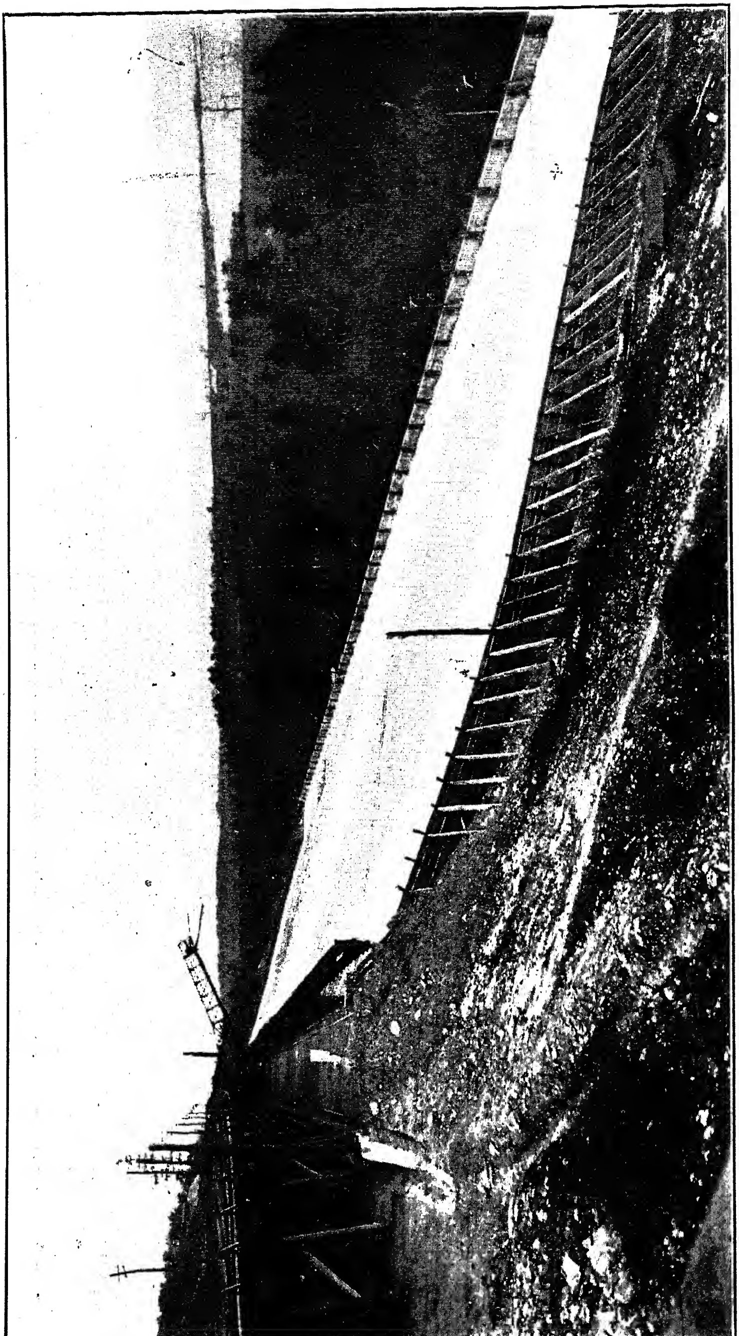

FIG. 9.—CONCRETE FLOOR, WITH CORRUGATIONS AGAINST RETAINING-WALL AT BOTTOM. HUDSONDALE STORAGE-PLANT.

and have yet to find the first one filling even approximately all the requirements. Where the surface was good there was usually no practicable means of approach, and where the pitch was suitable the line of face was impossibly irregular.

Given a not impracticable hill, a plant consists essentially of one or more dumping-tracks at the top, which in the older forms of plant, Fig. 5, are necessarily on rather high trestles, as it is self evident that no appreciable capacity could be obtained by dumping directly on the surface of the hillside, unless the slope was so steep as to make possible a pile thick at the bottom and tapering to nothing at the top, which would involve too great a drop and resulting breakage.

The coal is dropped from these trestles (the fall being broken as much as possible by chutes) and spreads down the hillside until arrested by walls, barriers, or by a level space at the bottom. It is evident that but little coal can be reloaded directly by gravity except the layer which may be held by a retaining-wall at the bottom, so it is usual to reload by hand, or better, by the use of dock-scrapers or swinging-conveyors along the level space at the bottom of the plant.

In one large plant almost all the coal is put into a conveyor at the foot of the hill and scraped to a central screen-house, where it is thoroughly rescreened and all the sizes recovered. In other cases reloading is done over fixed or shaking-screens placed at intervals above the tracks, and the screenings from these are taken by cars or conveyors to a small screen-house for separation.

In many cases the difficulty of handling at the foot of the hill is solved by the use of hydraulicking-water, best heated in winter, which is used under considerable pressure to carry the coal to the conveyors or cars for reloading, Fig. 5. This solves the problem of frozen coal as far as the storage-plant is concerned; but arrangements must be made for the disposal of the water, and in winter shipments the coal reaches its destination frozen to practically a solid block in the car, to the joy of the handlers at terminal points.

Where various sizes are stored it is necessary to provide partitions between the sections. These usually take the form of fences of heavy planking supported by large vertical posts, and braced by a forest of props, as shown in Fig. 5. The downward

motion of the coal has a strong tendency to dislodge these supports, with resultant heavy maintenance-cost. Moreover, to avoid admixture of dirt with the coal, it has been found necessary to protect the entire hillside, either by paving, planking, or concrete. This is particularly necessary where water is used in reloading.

The cost of installing a hillside storage-plant of this type is about $1.60 per ton of capacity complete, including railroads, trestles, partitions, water-supply, conveyors, screen-house, and planking. With concreted or paved hillside the cost would probably be a little higher.

The operating-cost, exclusive of fixed charges and deterioration of coal, but including labor, repairs, power, and shifting cars, will approximate 10 cents per ton for the coal passed through storage, dependent, as in all cases of storage-operating cost, on the activity of the plant.

The breakage of coal is somewhat large; the best figures obtainable for a plant of this type show the amount screened out of each size in reshipment, not the actual breakage determined by careful tests, as follows:

Size of Coal.	Breakage to Smaller Prepared Sizes	Breakage to Small Sizes.	Total Screened Out and Recovered.	Estimated Loss in Dirt.	Total Breakage
	Per Cent.	Per Cent.	Per Cent	Per Cent.	Per Cent.
Egg........	21.5	2.3	23.8	2	25.8
Stove........	8.8	2.7	11.5	2	13.5
Nut	......	13.6	13.6	2	15.6
Pea...........	......	12.5	12.5	1.5	14.0

The column "Estimated Loss in Dirt" is the fine dirt carried from the piles and from the screen-house by the water used in handling and preparation. At the plant in question, this dirt has been filtered out and forms a considerable waste-bank.

The above-mentioned figures were obtained from the regular operation of the plant, not from cleaned-up piles, and it is probable that an additional breakage will appear when the coal immediately under the trestle and at the bottom of the hill on the inside of the piles is cleaned up.

From the above it appears that the non-mechanical plants, types *a* to *f*, are generally expensive, both to erect and to

operate, do not generally lend themselves to the necessary screening, and involve a serious breakage of coal. On the other hand, they are suitable to small quantities of storage, lend themselves to separation of sizes and qualities, and are in general suitable rather to retail yards or the smaller type of wholesale piers than to extensive storage.

The line between the non-mechanical and the mechanical types is hard to draw, so many plants being combinations of both types. I have taken as mechanical storage all plants using machinery in storing coal, and as non-mechanical those storing by dumping, without regard to the occasional incidental use of machinery for reloading in some of the non-mechanical plants above described.

2. *Mechanical Storage-Plants.*

(*g*) *Hillside with Mechanical Stocking.*—The most notable plant of this type was constructed during 1905–6, for the Lehigh Valley Coal Co., at Hudsondale, Carbon county, Pa., under my supervision.

Owing to the high breakage-loss in prepared sizes in hillside storage, it was considered inadvisable to use this type for prepared coal, and the plant was designed and is operated exclusively for the storage of small sizes.

The situation, on the Quakake branch of the Lehigh Valley railroad, is on the line of haulage from the Schuylkill district to tide-water, within a couple of miles of the junction of the Lehigh branch, and only about 10 miles back from the main line of the Lehigh Valley railroad, a satisfactory point for tide-water deliveries, far enough from the mines to avoid interterence, and yet minimizing the capital locked up in freight-charges on stored coal.

The hillside selected was fairly straight and true in grade, but required heavy earth-work for the reloading-tracks, and the stocking-track at the head of the hill was inaccessible at reasonable grades without prohibitory cost, and is reached by an engine plane.

The plant, Fig. 6, differs from all previous hillside plants in many particulars. Owing to the configuration of the ground the loaded cars are hoisted up a plane 500 ft. long on a 30-per cent. grade, by a pair of 18- by 30-in. hoisting-engines, double

geared 16 to 1 to a 10-ft. drum. The cars are pushed up by a steel barney, which returns into a pit at the foot of the plane. From the head of the plane the cars run over a double-track

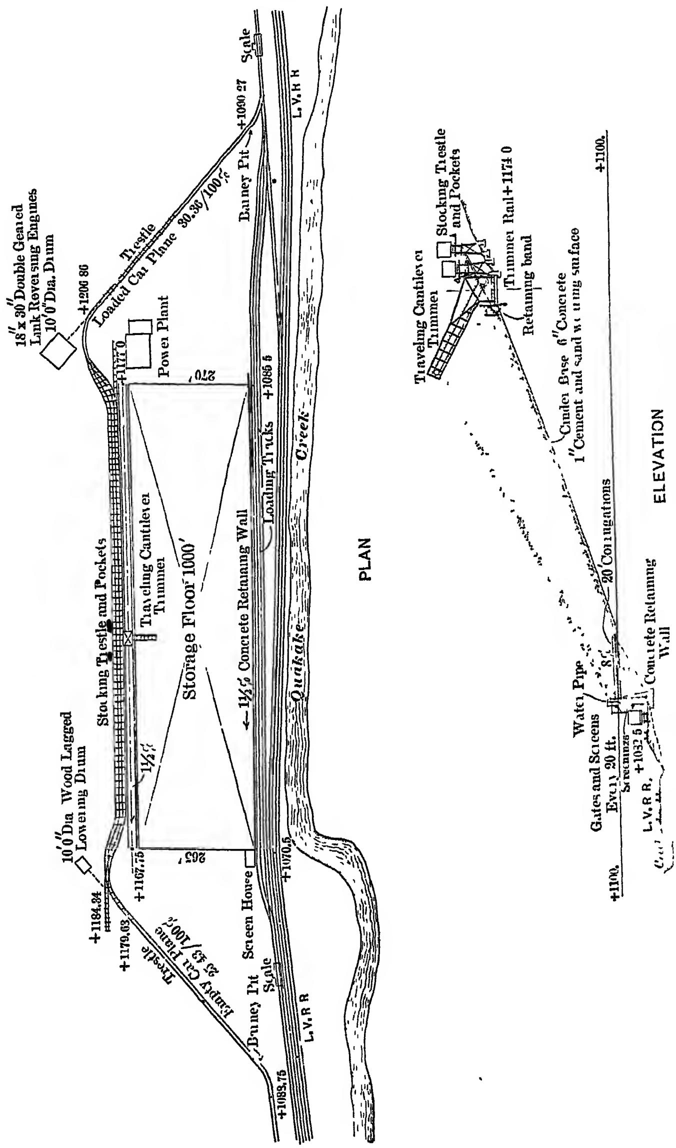

FIG. 6.—HILLSIDE STORAGE-PLANT, LEHIGH VALLEY COAL CO., HUDSONDALE, PA. PLAN AND ELEVATION.

trestle, shown in Fig. 6, just high enough to permit dumping the coal into a traveling-cantilever trimmer, Fig. 7, by which it is elevated and discharged on to the concrete-floored hillside,

making a pile more than 55 ft. deep at its maximum, tailing down against a concrete retaining-wall extending 7 ft. above the storage-floor. This wall, Fig. 8, has a total height of 24 ft. above the reloading-tracks, and is provided with openings on 20-ft. centers discharging the coal over screens directly into railroad-cars for shipment. The screenings are washed in a trough to a small screen-house at the lower end of the plant, where they are rescreened for shipment. As but a small portion of the coal is accessible by gravity, the main reloading is done by the use of water pumped from a nearby creek to a storage-tank on the hill above the plant, and used with hose-streams to wash the coal to the gates and over the screens.

Railroad-cars are handled by gravity on both reloading- and stocking-tracks, and the empty cars from the latter are lowered on a plane, operated by a drum with powerful air-brakes, to the level of the railroad.

Except the hoisting-engines for the loaded-car plane, the entire plant is electrically operated and lighted from a station included in the equipment.

The entire cost including all charges approximated $1.50 per ton of capacity, and when in active operation the handling-cost has reached the record figure of 1.25 cents per ton handled through the plant.

The many unique features of this plant, which is considered an advance on all previous plants of this type, merit further detailed description.

The two tracks on the dumping-trestles, Fig. 6, are at different elevations, to minimize the drop at this point, and the chutes under these form a shallow pocket controlled by numerous gates. This pocket, while not of a depth to increase the drop from the hoppers of the cars, has sufficient capacity to give the trimmer a continuous supply, regardless of the variations in discharge in unloading and moving the cars.

The cantilever trimmer, the invention of S. D. Warriner, Vice-President and General Manager of the Lehigh Valley Coal Co., shown in Figs. 6, 7, 8, and 9, was designed and built by the Link Belt Co., of Philadelphia. It consists of a platform traveling parallel to the dumping-trestle on a 16-ft.-gauge track and carrying a cantilever-truss equipped with a scraper-conveyor. The bottom of the conveyor-trough is mov-

able, so that the point of discharge can be at any desired place. To increase the capacity of the plant and avoid stored coal flowing back on to the trimmer-tracks, a bulkhead anchored to a retaining-band in the coal, Figs. 7, 8, and 9, separates the trimmer-track from the storage-floor. Except the drop from the cars to the chutes immediately below and just clearing the hoppers, the only other drop involved in storing coal is in making the first small pile behind the bulkhead. After this reaches the line of trimmer the pile is built by moving the discharge outward, and the coal from the end of the trimmer reaches the growing pile without appreciable drop, and extends the pile by avalanching, as previously described.

The storage-floor averages 260 by 1,000 ft. on the hillside. This was first trued to squares 25 ft. on a side, so designed as to give the best slopes without re-entrant angles attainable without too serious grading. The floor thus prepared was covered with from 2 to 3 ft. of cinders, placed by the use of a temporary traveling cable-way, and then with 6 in. of cinder-concrete with a wearing surface of 1 in. of cement and sand. The entire preparation of the floor cost a little less than 26 cents per square foot, of which nearly 14 cents was for the concrete. The lowest 30 ft. of the floor is on a much flatter grade than the rest, and with a view to a better conduction of the water and coal over this section the floor is made with 20-ft. corrugations, the bottom of each leading to a gate, Fig. 9. Experience has proved the advantage of this arrangement, and further, that it would have been very advantageous to carry these corrugations the entire width of the floor, as considerable difficulty is encountered in washing down the fine coal by reason of the spreading of the water. In many cases in reloading coal temporary iron chutes are laid to prevent this spread.

The retaining-wall, Fig. 8, was built of concrete reinforced with old wire rope, with an aggregate of crushed mine-refuse; this, by reason of its character, has somewhat deteriorated the concrete, and the wall, while designed amply against overturning, and anchored back by numerous tie-rods, has been forced forward to some extent in places, probably by the freezing of water in the fill behind it.

The problem of letting down the loaded cars was solved by the use of a second plane, single track, with a barney ahead of

the cars disappearing at the bottom into a pit. The controlling-drum lowers by means of a band-brake on an asbestos-lined brake-wheel, operated by a standard Westinghouse air-brake equipment, supplied with air by an automatic electrically-driven air-pump. The barney is hoisted by a small motor, clutch-connected to a train of gearing operating the drum, and runaways are guarded against by a governor, which sets the brake in case a safe speed in lowering is exceeded. The brake is also arranged for hand-operation in emergency.

Different sizes when stored are either separated by temporary bulkheads of the type shown in Fig. 21, or the edges of the piles are allowed to mix, the sizes being separated by the shipping-screens.

As this plant is used (and is suitable) only for the small sizes of coal, the question of breakage is not of supreme importance, and no accurate figures are available as to its amount. From observation, I would consider it small, probably not much exceeding that in a standard Dodge plant.

(*h*) *Traveling or Fixed Tramway Storage.*—The tramway type of storage, stocking and reloading by traveling-buckets, while in very general use for ore-storage, has been but little used for stocking anthracite on an extensive scale, largely on account of excessive breakage, the impracticability of rescreening before reshipment, and small handling-capacity.

The largest plant of this type for anthracite storage was built for Coxe Bros. & Co., at Roan Junction, Pa., with a capacity of 100,000 tons in a continuous pile, since increased to more than 150,000 tons.

This plant, Fig. 10, consists essentially of a traveling-truss, 225 ft. span, with 100 ft. cantilever-extension and 40 ft. back-projection, built by the Brown Hoisting Machinery Co. The truss is 55 ft. high above the rail at the traveler, and the bottom member has an elevation of 40 ft. above the storage-ground. The truss is supported by a tower, spanning the reloading-tracks and containing the engines and boiler for operation. The outboard end, supported by an A-frame, travels on a single rail, outside of which the stocking-track is elevated to such a height that cars can be dumped into small hoppers, 50-ft. centers, Fig. 11, from which the coal is drawn into 5-ton buckets, supported on a traversing-truck. One bucket is hoisted, carried

along the truss, lowered, and dumped on the stock-pile while its companion is being filled; these buckets dump automatically only when resting on the stock-pile.

The area covered by the stock is 200 ft. wide between the stocking- and reloading-tracks, and 100 ft. beyond the latter, the center of this area being reached by the cantilever-extension. The storage-area, originally 1,200 ft. long, has now been extended to 1,550 feet.

Reloading is accomplished by the use of a 3-ton "shovel-bucket," Fig. 12, which is filled by pulling it over the surface of the coal, and dumped by hand into cars at the reloading-tower.

Attempts have been made to rescreen the coal in shipment by the use of a traveling screening-pocket attached to the tower, but with such little success that rescreening at this point has been abandoned.

While a large storage at low cost per ton is attained, the handling-capacity of the plant is small, the average rate of stocking is but 83 and of reloading 70 tons per hour, woefully insufficient for a plant of this capacity. This condition could, of course, be remedied by the use of several trusses, which, however, would greatly increase the cost of installation.

The breakage, particularly in reloading, is heavy, and on this account the plant is chiefly used for the smaller sizes. The original cost of construction is said to have been but $60,000, or 60 cents per ton of rated capacity. The present cost would be at least 50 per cent. greater. The cost of operation averages slightly over 5.5 cents per ton for stocking and about the same amount for reloading on a total exceeding 150,000 tons handled, including all labor, repairs, and train-service, but not interest-charges or depreciation of plant.

An interesting plant of this type is situated at Fall River, Mass., Fig. 13, where I met the problem of unloading coal from barges, transferring it either to railroad-cars, stock-yard, or retail-pockets, and reloading from the stock-yard to either pockets or cars.

The present plant replaced one consisting of hoists, which dumped into transfer-barrows, thence, by a weighing-hopper, into cable-cars on a trestle surrounding the plant, finally dumping into the storage-yard or into the pockets. Trans-

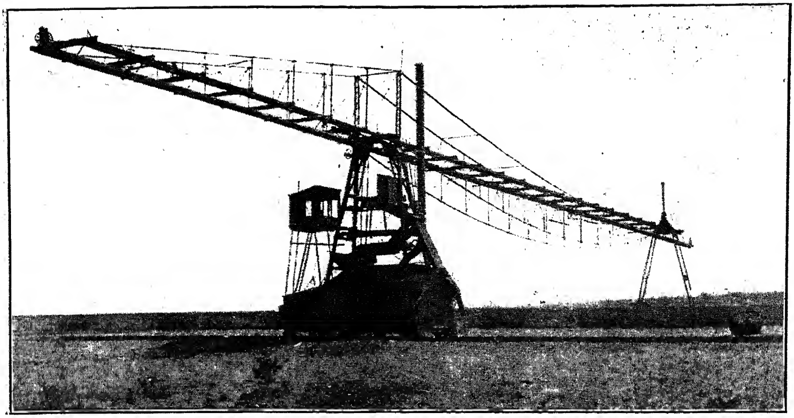

FIG. 10.—TRAVELING TRAMWAY, COXE BROS. & CO., ROAN JUNCTION, PA.

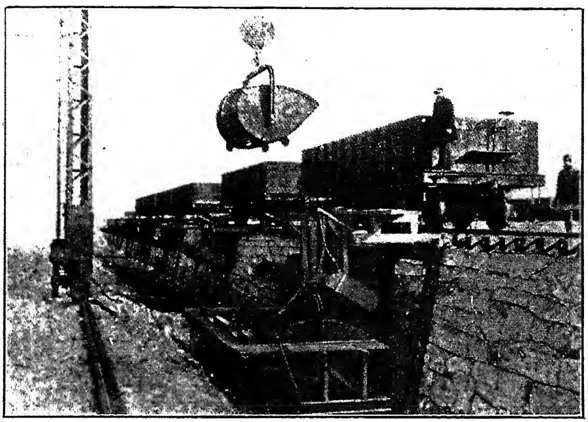

FIG. 11.—STOCKING-TRACK, ROAN JUNCTION, PA.

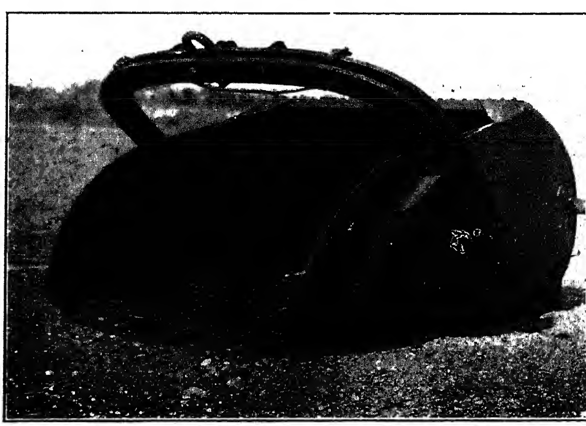

FIG. 12.—SHOVEL-BUCKET FOR RELOADING, ROAN JUNCTION, PA.

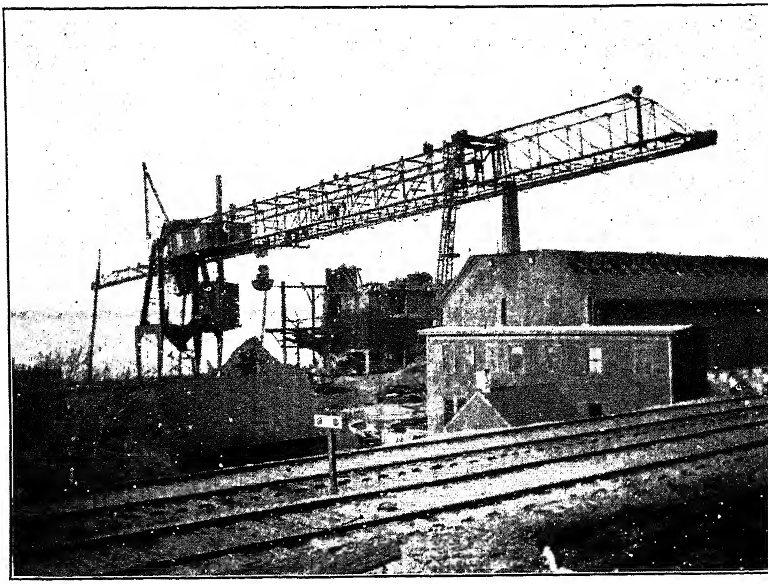

Fig. 14.—Traveling-Tramway Storage- and Handling-Plant, Fall River, Mass. Showing Pockets and Old Plant in the Background in Process of Demolition.

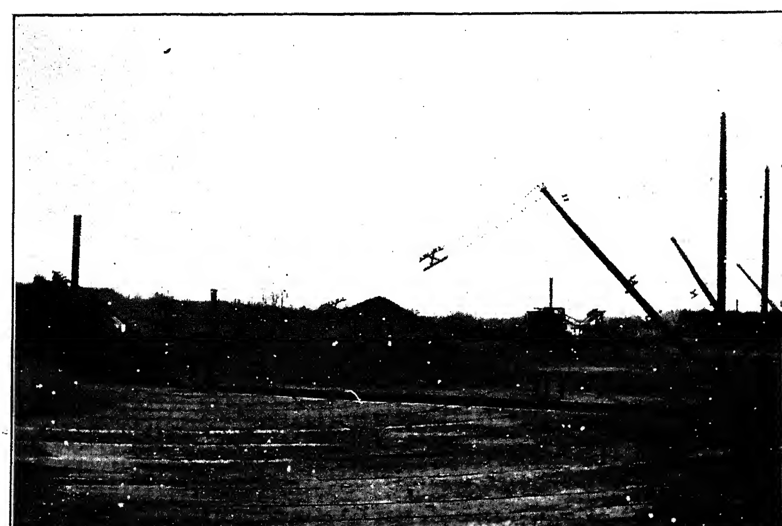

Fig. 16.—Original Type of Dodge Plant, Pennsylvania Railroad, South Amboy, N. J. With Mast and Boom Supporting Trimming-Conveyor.

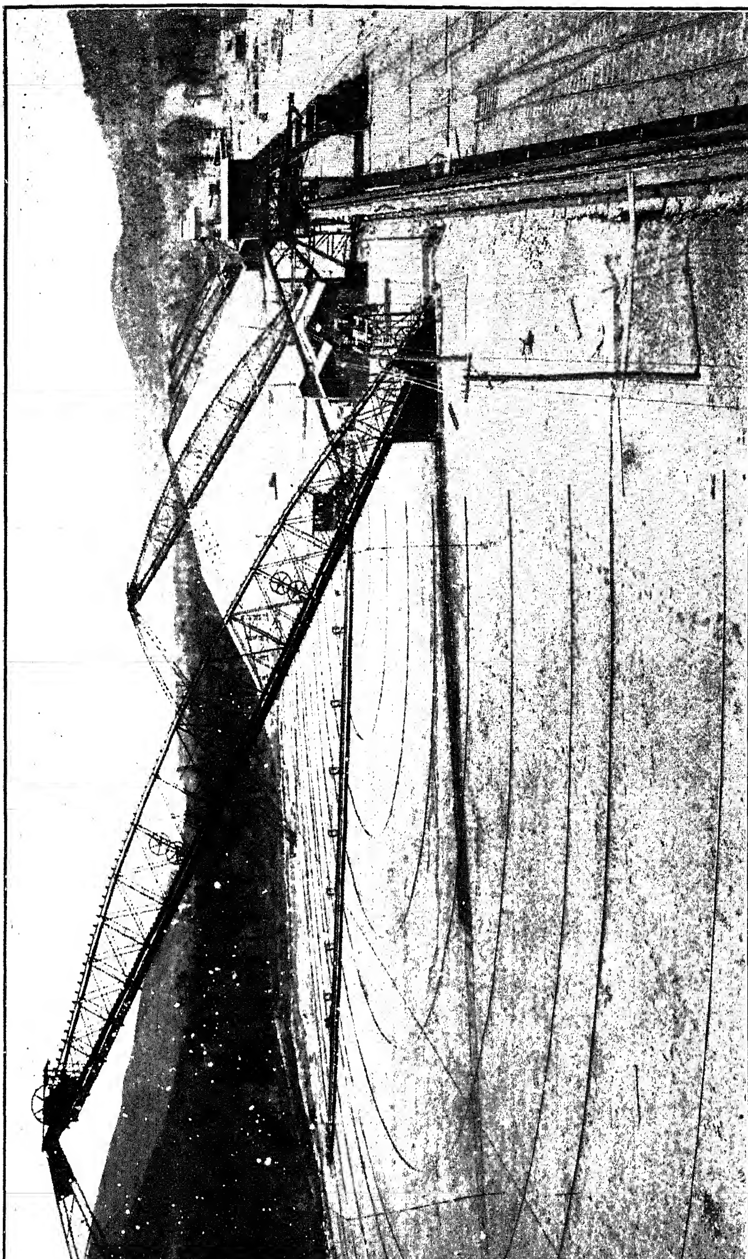

FIG. 17.—STANDARD DODGE PLANT, HAUTO, PA. SHOWING TRUSSES, RELOADING-TOWERS, AND SWINGING-RELOADER.

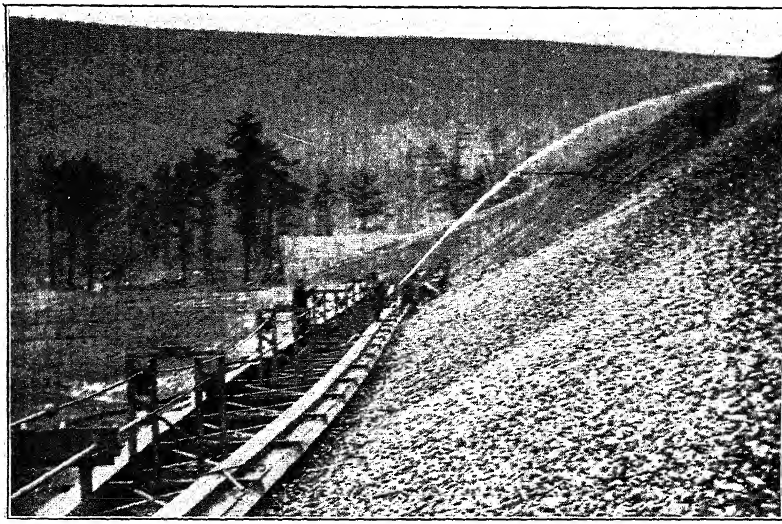

FIG. 18.—RELOADER WORKING ON PILE OF COAL, HAUTO, PA.

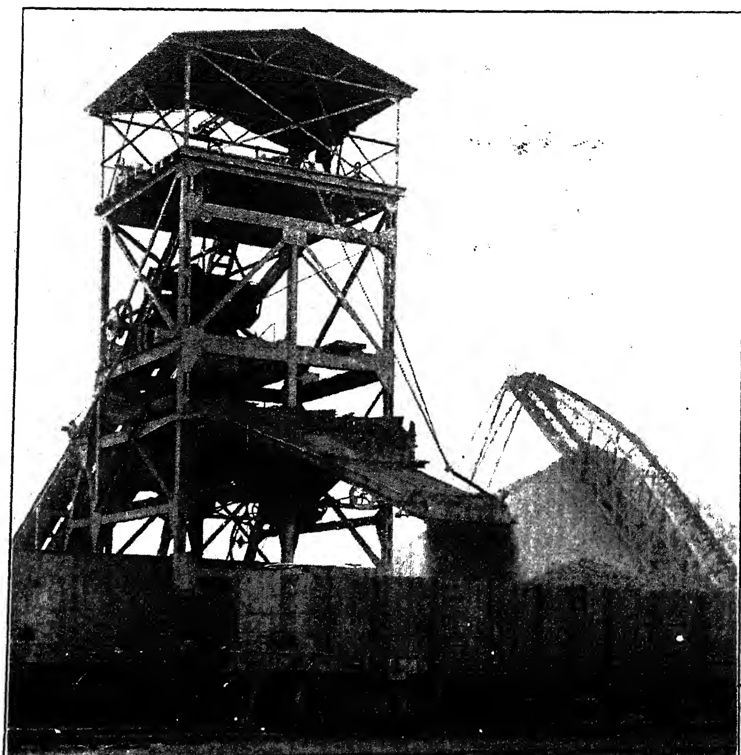

FIG. 19.—RELOADING-TOWER AND SHAKING-SCREENS, ABRAMS PLANT, P. & R. C. & I. CO.

shipment to the railroad was accomplished by separate pockets, and any reloading from stock was done by hand. The original plant lost about 7 per cent. in breakage screened out, and cost 18 cents per ton for handling. But the 7 per cent. by no means covered the entire breakage, as only inefficient screening was done, and merely fine dirt removed.

The new plant consists, Fig. 14, of a traveling-tramway, with cantilever-extension over the pockets and hinged-bridge

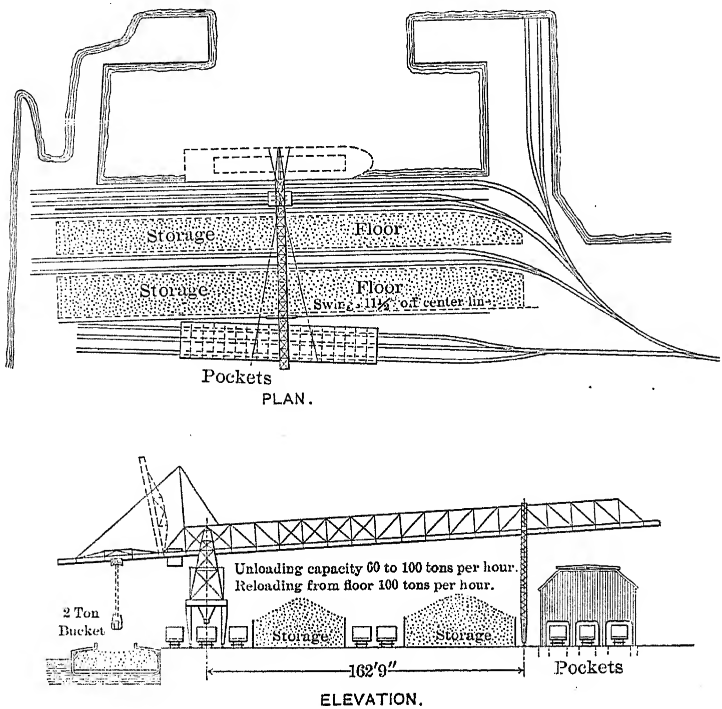

Fig. 13.—Traveling-Tramway Storage- and Handling-Plant, Staples Coal Co., Fall River, Mass. Plan and Elevation.

extension to extend over the barges. The tramway, built by the Dodge Coal Storage Co., is hung from its supports by a number of thin eye-bars, giving flexibility sufficient to permit of swinging 11.5° either side of the center-line, allowing a variation of 50 ft. each way over the pockets, which is necessary to permit of the selection of pockets for various sizes of coal.

Unloading, both from the barges and from stock, is done by

means of a 2-ton clam-shell bucket, in which coal is carried to the desired point, lowered, and let out either on the storage-pile or in the pockets, which are large enough to receive it. Trans-shipment from water or stock to the railroad is accomplished by the same bucket, discharging into a steel hopper in the tramway-tower, and thence, by gates, to the cars below.

The plant handles both anthracite and bituminous coal, as may be required, and in reloading from stock the tramway is assisted by a locomotive-crane with clam-shell bucket of 0.5-ton capacity.

The cost of operation in the plant has been reduced to about one-third of its previous cost. The total cost of the plant was about $50,000, and the saving by its use exceeded 10 cents per ton on 150,000 tons handled per year, besides reducing the screenings from 7 to less than 4 per cent.

The guaranteed speed of operation is 100 tons per hour, which rate in practice has been nearly doubled in emergency.

In general, the tramway system, within its limitations, is probably the lowest in first-cost of all the storage-systems, while the operating-cost is between that of the non-mechanical and the large mechanically-operated plants. The principal advantages of this type are low first-cost, flexibility, moderate labor-cost and repairs; the disadvantages, large space occupied by reason of relatively low piles, danger from wind, excessive breakage, unless very carefully handled (from the tendency of the operators to dump the buckets without lowering to the stock-pile), and lack of facilities for rescreening in loading out from stock.

(*i*) *Dodge Storage-System.*—The Dodge system with its modifications is used for anthracite storage probably more extensively than all others combined. This system, invented by James Mapes Dodge, of Philadelphia, fills more nearly than any other the conditions of an ideal plant. In its standard form, Fig. 15, anthracite is stored in conical piles by means of a trimmer-truss carrying a flight-conveyor, with a movable bottom, which discharges at the apex of the growing conical pile, and reloading is accomplished by a horizontal swinging-truss, placed between two conical piles, carrying on its edge a flight-conveyor. This conveyor takes the coal from the edge of the conical pile, draws it to a central point, and by a change

in direction carries the coal up an incline to a tower, in which it is thoroughly screened on its way to the car.

The earliest large plant of this type was built for the Pennsylvania Railroad Co., at South Amboy, N. J., in 1889, with a capacity of 100,000 tons. In this crude plant, Fig. 16, the upper end of the trimming-conveyor was supported by a boom projecting from a wooden mast erected back of the center of the pile, and the reloader was traversed by hand and delivered into a pit, whence the coal was elevated for shipment, no rescreening being attempted.

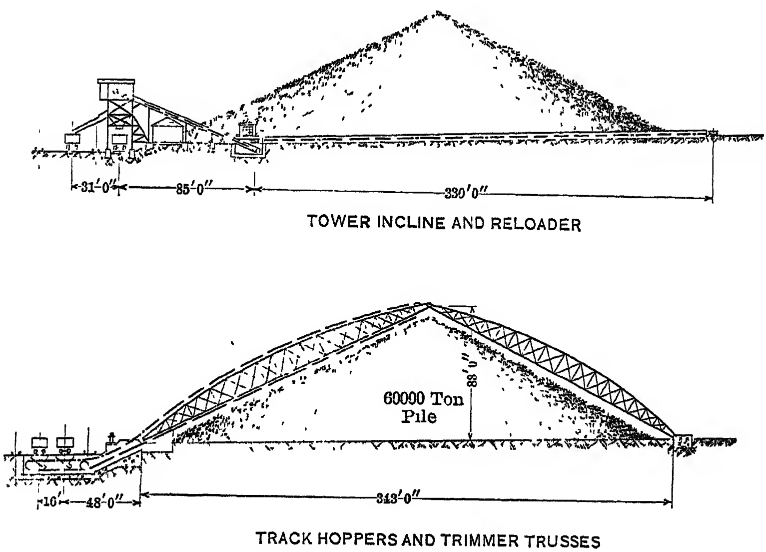

FIG. 15.—STANDARD TYPE OF DODGE STORAGE-PLANT.

In the modern plants of this type, Fig. 15, the trimming-conveyor is supported by a light hinged arch-truss, Fig. 17, of span suited to the size of the pile, with a pitch equal to the angle of repose of the coal, carrying in its bottom member the trough-and-chain conveyor, which returns over the top. The bottom of the trough is a single movable strip of sheet steel wound on a drum at the foot of the truss and pulled by power up the truss, advancing as the pile grows, leaving an open bottom above the point of discharge, thus minimizing the breakage at this point, as the coal is merely shoved out on to the point of the conical pile and builds the pile by avalanching rather than by rolling. The thrust of the arch-truss is taken

up by tie-rods extending under the storage-floor, and wind-pressure is provided for by guy-ropes extending above the surface of the coal to anchorages outside the piles. The trimming-conveyor extends from the foot of the truss on a catenary curve to an extension under the dumping-tracks, where hoppers are provided, feeding the conveyor to capacity by adjustable gates.

Two trimming-trusses with respective track-hoppers and a central reloader form a unit of construction.

The reloader, Figs. 17 and 18, is pivoted between the two piles, and swings on curved supporting-tracks, just clearing the outer ends of the trusses, and covers both floors, leaving only a small crescent-shaped pile outside its reach on each floor. These piles are handled either by hand or by dock-scrapers to within reach of the end of the reloader. The reloader-truss, carrying the moving conveyor on its faces, is fed by power against the bottom of the pile, being operated from a station on the pivot, from which a full view of the operation is assured. As the piles cone down by avalanching, and not by continuous rolling, it is often necessary to back out the reloader in a hurry to avoid having it buried. The movement is accomplished by wire cables which lie along one of the circular tracks under the coal, and the ends of which coil on reversing-drums in the engine-house, controlled by clutches from the operator's platform.

At the pivot-end of the reloader the chain carrying the conveyor-flights is deflected up an incline to the reloading-tower, Fig. 19. In the case of the largest piles, the strain from this extension has proved too great for the Dodge chain necessarily employed in making this turn, and separate conveyors are installed on the reloader and tower. The reloader-conveyor in this case transfers to the tower-conveyor.

The reloading-tower contains shaking-screens of ample capacity to rescreen the coal fully, and after passing over these the coal goes by a chute to the cars for reshipment. These loading-chutes are long and originally caused considerable breakage, but the later ones are covered and provided with an end-gate, by means of which the chutes can be kept full and the coal poured from the end without the velocity which would be acquired in a free slide for the length of the chute.

The screenings are collected in hoppers in the towers, and in modern plants they are taken to a separate screen-house for re-preparation into marketable sizes, either by long conveyors or by cars, with rope- or locomotive-haulage.

Power is provided for the operation of each unit from engines or motors in a house adjoining the reloading-tower. The trimmer-conveyors, while occasionally driven by motors at the top of the trusses, are usually operated by rope-drives from the engine-house to the head-sheaves on the trusses, with the object of minimizing the weight on the truss.

It is evident that but one size and kind of coal should be stored in any one pile, and this limitation, involving the installation of numerous piles, is the most serious objection to the system.

The approximate cost of the machinery and trusses, per ton of capacity, varies greatly with the size of unit-piles.

Approximate Cost of Dodge Anthracite Storage Groups.

Capacity	Units.	Diameter.	Height.	Cost Installed.	Cost Per Ton.
Tons.		Feet	Feet.		
120,000	2- 60,000	333	85	$79,500	$0.6625
100,000	2- 50,000	313	80	72,000	0.72
80,000	2- 40,000	293	74	65,000	0.8125
60,000	2- 30,000	263	67	59,800	0.995
50,000	2- 25,000	248	63	53,900	1.08
40,000	2- 20,000	230	58½	50,600	1.265
30,000	2- 15,000	208	53	46,200	1.54

To this amount must be added the cost of foundations, track-hopper pits, preparation of floors, central power-plant (steam or electricity) and power-distribution, drainage, screen-house for screenings, and railroad-tracks, scales, and yards.

The most modern plants have been built of great capacity, with large unit-piles of from 50,000 to 60,000 tons capacity, with the result of reducing the first-cost of a complete plant from $1.50 per ton of capacity for a 300,000-ton plant, with 25,000-ton units, to $1.06 per ton for a 480,000-ton plant, with 60,000-ton units.

Depending upon the size of units, the handling-capacity varies from 50 to 150 tons per hour for stocking or reloading, which speed is attained easily in actual work, including the time lost in spotting and opening the hopper-bottom steel cars.

Owing to the thorough rescreening in use, the breakage in handling by this type of plant is quite accurately known. In the operation of a typical modern plant the following breakage-calculation from cleaned-up piles has been recorded.

Amount Screened Out as Smaller Sizes.

Size Stocked	Stove.	Nut.	Pea	Buckwheat	Rice, Barley and Dirt.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Egg	8.9	2.4	0.58	0.50	1 82
Stove	......	3.9	0.93	0.65	0.37
Nut	......	......	1 40	1.10	0 36
Pea	......	......	.....	1.01	0.37
Buckwheat	.. ...	.. ...	......	.... .	0.56

I think that this fairly represents the breakage, as the coal received contained some undersize, apparently in about the same quantities and proportions as that shipped.

This loss, figured on 1,000,000 tons of assumed quantities of each size passing through storage, is:

Size	Original Quantity	Price Per Ton.	Original Value	Quantity Shipped	Price Per Ton.	Value
	Tons			Tons		
Egg	350,000	$5.00	$1,750,000	300,300	$5.00	$1,501,500.00
Stove	200,000	5.00	1,000,000	219,450	5.00	1,097,250.00
Nut	200,000	5.00	1,000,000	210,480	5.00	1,052,400.00
Pea	250,000	3.25	812,500	253,240	3.25	823,030.00
Buckwheat	....	....	. .. .	7,775	2.50	19,437.50
Rice / Barley	....	....	....	8,755	1.75	15,321.25
Total	1,000,000	....	$4,562,500	1,000,000	....	$4,508,938.75

Loss, $53,561.25 ; or 5.36 cents per ton.

The cost of operation, fairly averaged at 5 cents per ton handled each way, is extremely variable, dependent upon the activity of the plant. For a large tonnage it has been as low as 2.4 cents per ton, and for three consecutive months it averaged 4.6 cents per ton, including all labor, repairs, and supplies, but not interest, taxes, or depreciation, with occasional jumps to 35 cents or 40 cents per ton during inactive times when but little coal was handled and the fixed charges for attendance dominated the cost.

An essential feature of this type of plant is ample railroad-

trackage. Two tracks for dumping and reloading, one service-track, and one track for screenings, with ample cross-overs, are none too much for a single line of piles. Owing to the large handling-capacity of each unit, and the necessary number of units to provide for the various sizes and kinds of coal, the total handling-capacity of the plant is enormous, and for busy times it does not seem that too much trackage can be provided. The tracks through the plant operate by gravity best on a 1.25-per cent. grade through the yard, stiffened to 1.5 per cent. over the dumping-hoppers and in front of the reloading-towers, and reduced to 1 per cent. in the loaded classification-yard, which is required below the plant. A plant of 500,000 tons capacity will be nearly 1.5 miles long, and will contain in the aggregate about 10 miles of tracks. Where the contour of the ground does not lend itself to gravity-handling of the cars, a wire-rope haulage, with very slow speed of operation, is usually installed for this purpose.

The power required to operate a plant of this type was determined for a 60,000-ton unit, two 30,000-ton piles, at the McClellan plant of the Susquehanna Coal Co., to be:

	I H-P.
Engine and attached machinery, light, ...	15.5
No. 1 trimmer-conveyor, empty, ...	37.0
No. 1 trimmer-conveyor, loaded, ...	53.5
No 2 trimmer-conveyor, empty, ...	36.7
No. 2 trimmer-conveyor, loaded, ...	53.3
Reloader-conveyor, empty, ...	38.7

In the screen-house and on the towers, each shaking-screen, 6 by 12 ft. in size, required 2.62 h-p. for operation.

At the time when this test was made reloading was not in progress, so no test could be made on the reloader actually in service.

The most recent plant of the standard Dodge type was erected in 1907–08, for the Lehigh Coal & Navigation Co., at Hauto, Pa.

The detailed costs of this plant are available through the courtesy of W. A. Lathrop, President, and Baird Snyder, Jr., General Superintendent of the company.

The plant, Fig. 17, consists at present of four 30,000-ton and

two 60,000-ton piles, total capacity 240,000 tons, arranged in line on one side of the tracks, the other side being reserved for extensions. At the present time two more 60,000-ton piles are being erected, increasing the capacity to 360,000 tons, which should be available early in the summer.

Special features of the plant are electrical driving from the central station of the Lehigh Coal & Navigation Co., at Lansford. Each unit, two piles with pivoted reloader, is driven from its own power-house; the transmission to the trimmers, reloader, and loading-tower of each is by rope-drives. Each loading-tower is equipped with a shaking-screen, 5 by 12 ft. screening-surface, provided with a full set of perforated plates for any size of coal. The screenings are washed in troughs to a very complete screen-house at the lower end of the plant. Sufficient grade for this washing is obtained by the use of two elevator-towers in the line of troughs, which by raising the screenings avoid undue elevation of the troughs.

The screen-house is provided with breaking-down rolls and a full set of screens for separating the screenings into sizes, which are shipped directly from the screen-house pockets.

The site selected is a favorable one for this type of storage. No excessive grading was required, and drainage is available, so that it is the practice to use water for reloading frozen coal, Fig. 20.

As in all plants of this type, the capacity of the piles is rated on the assumption of strictly conical structure, built directly by the trimming-conveyors, while in case of necessity the piles can be materially extended by the use of sheet-iron chutes from the head of the trimmer. In this plant such extension has been carried to the limit by the further use of plank bulkheads between the piles, Fig. 21, so that a rated 30,000-ton pile of egg-coal actually contained 70,600 tons, more than 135 per cent. above its rated capacity. The bulkheads are built with a face of 2-in. plank, retained by cleats of plank extending into the body of the coal and held against spreading by the friction of the coal itself.

The cost of the present 240,000-ton plant complete was \$415,771.70, or \$1.732 per ton of rated capacity, made up of items as follows:

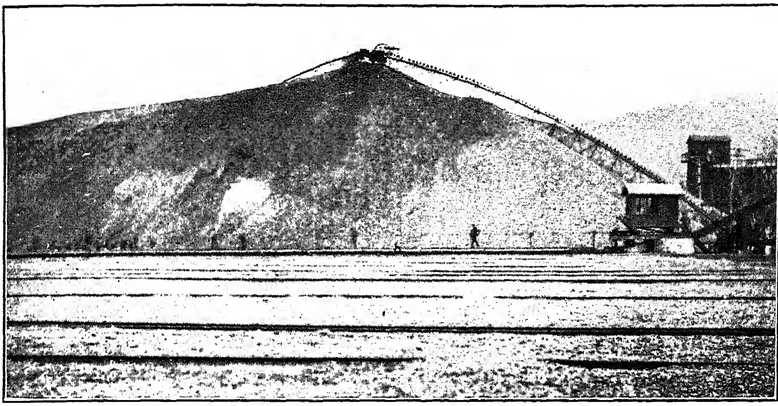

FIG. 20.—HAUTO PLANT. 30,000-TON RATED CAPACITY PILE, WHICH CONTAINED 70,600 TONS EGG-COAL, IN PROCESS OF RELOADING, SHOWING USE OF WATER FOR THAWING FROZEN COAL.

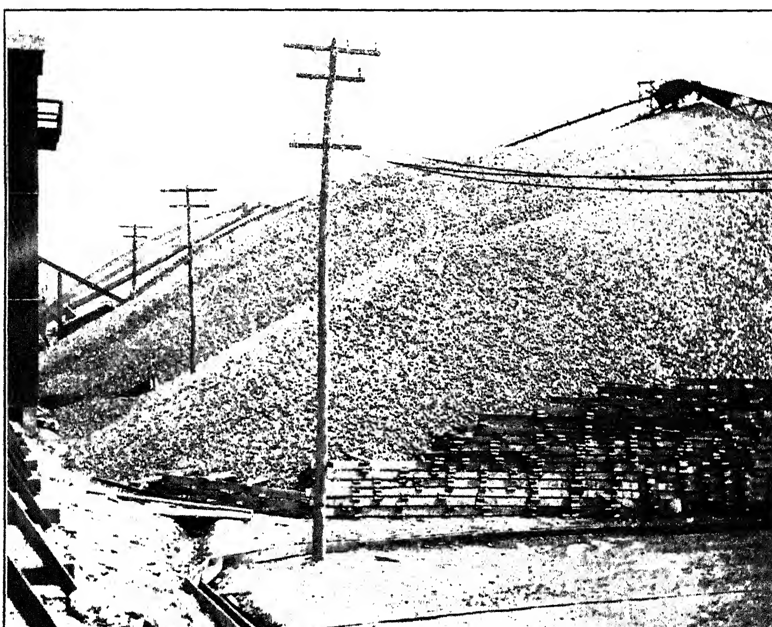

FIG. 21.—HAUTO PLANT. TEMPORARY PLANK-BULKHEAD FOR RETAINING PILED ANTHRACITE.

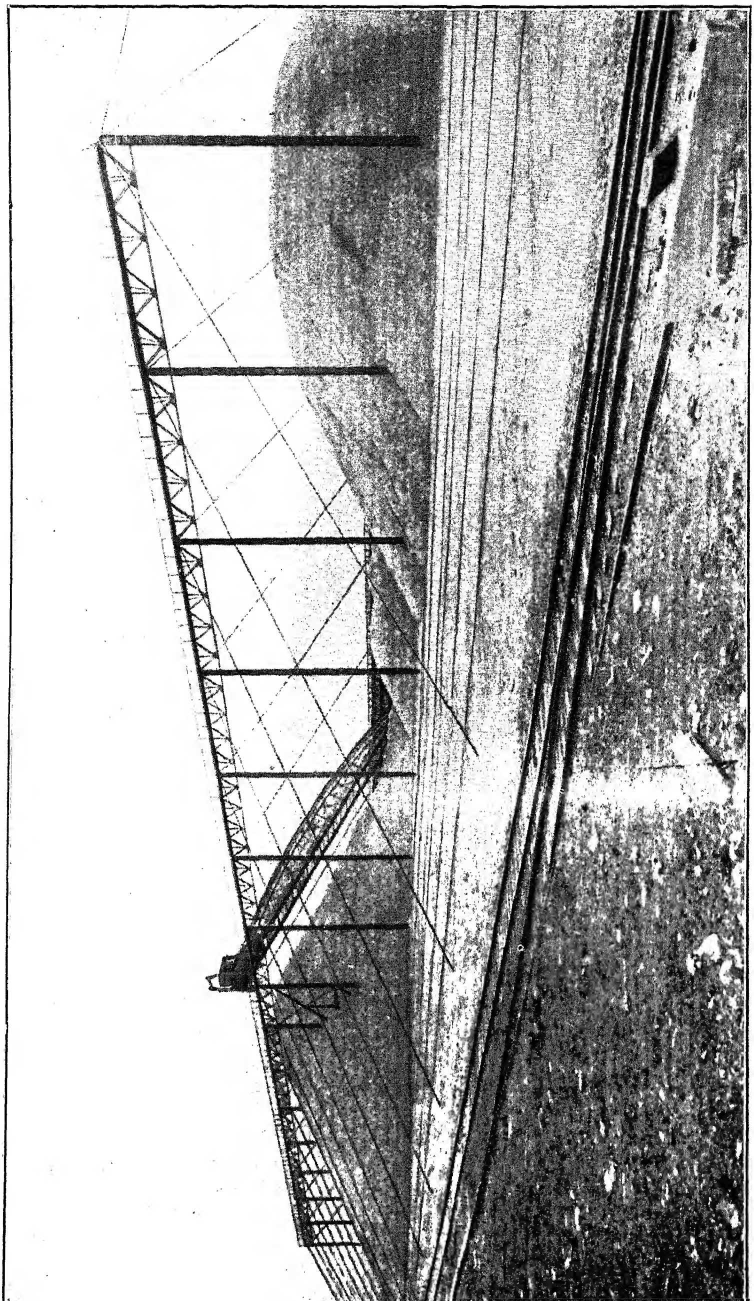

Fig. 23.—Ransom Plant. General View of Colonnade and Traveling-Trimmer.

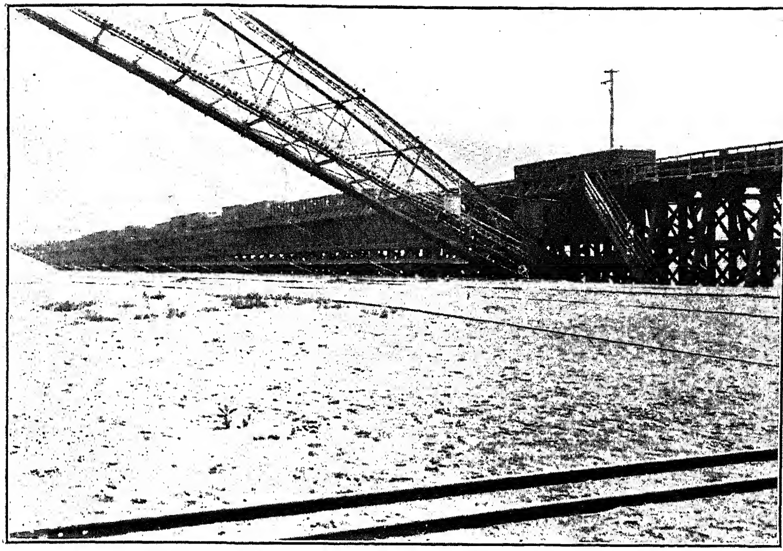

Fig. 24.—Ransom Plant. Stocking-Trestle, Showing Bins, Chutes, and End of Traveling-Trimmer.

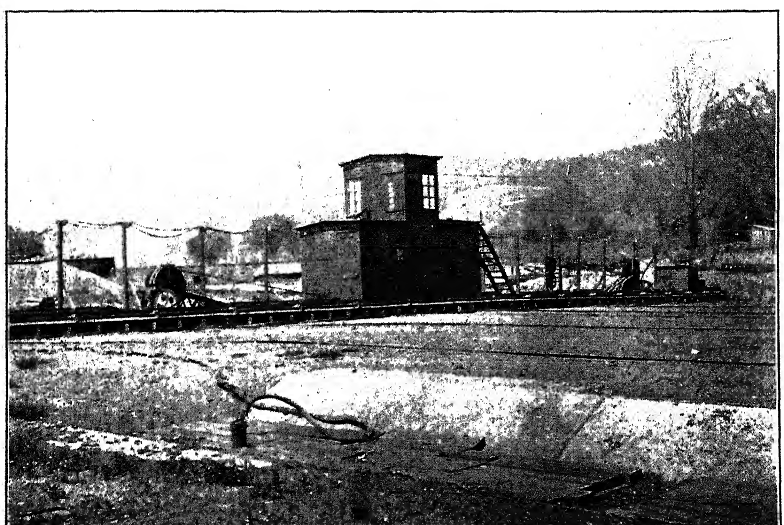

Fig. 25.—Ransom Plant. Traversing-Reloader, Showing Electrical Connection, and Hopper-Slot Plank and Covering of Longitudinal Tunnel.

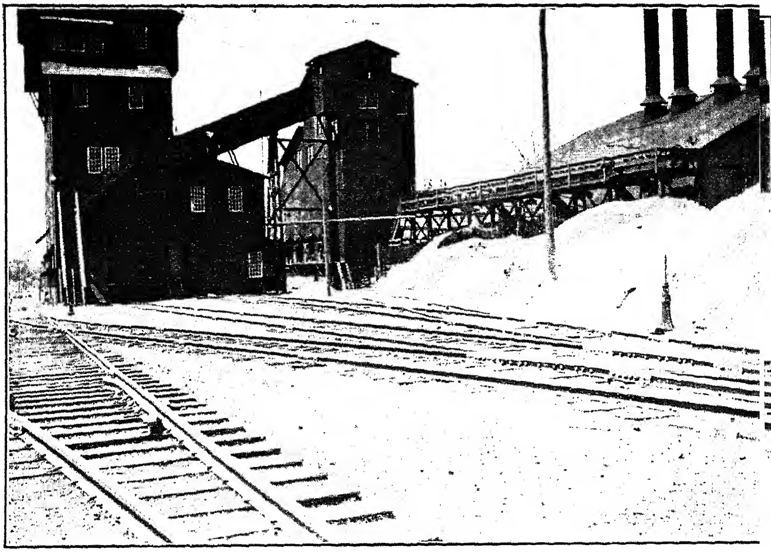

Fig. 26.—Ransom Plant. Reloading-Tower, Screenings-Separator, and Power-Plant.

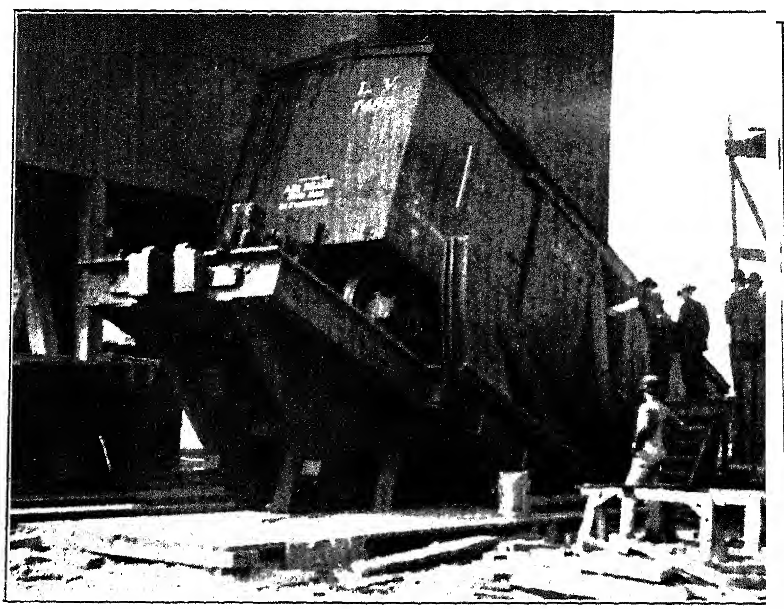

Fig. 27.—Smith Box-Car Reloader. Ransom Plant.

		Per Ton of Rated Capacity
Grading and masonry,	$94,996.49	$0.395 —
Railroads,	32,656.84	0 136
Buildings,	26,070.54	0.108 +
Machinery,	215,766.73	0.900 —
Electric installation,	15,829.81	0.066
Screen-house,	28,415.37	0.119 +
Electric power-transmission,	2,035.92	0.008
	$415,771.70	$1.732

The two 60,000-ton piles now under contract are estimated to cost $120,000, which will make the entire cost of the 360,000-ton plant $536,000, or $1.49 per ton of rated capacity.

The cost of operation for the first year only is available, amounting on 209,690 13/20 tons handled to $9,263.59, or $0.0442 per ton, as follows:

	Amount.	Cost Per Ton
Superintendence,	$584.62	$0.00279
Labor,	3,541.48	0 0169
Supplies,	1,536.39	0.00732
Repairs,	80.68	0.0003
Electric power,	1,133.67	0.0054
Cost,	$6,876.84	$0.0328
Transportation,	2,386.75	0.01143
Total cost,	$9,263.59	$0.0442

With the excellent rescreening facilities provided, coal is shipped in condition fully up to the standard of breaker-shipments, and the breakage due to storage was accurately determined from the repreparation of the screenings, except that no size larger than chestnut was taken out in rescreening, leaving all stove-size in the egg as shipped.

The degradation from cleaned-up piles has been as follows:

Sizes Stocked.	Nut.	Pea.	No. 1 Buckwheat.	Rice.	Barley and Dirt.	Total Below Prepared Size
	Per Cent.	Per Cent.	Per Cent.	Per Cent	Per Cent.	Per Cent.
Egg	1.47	0.681	0.42	1.047	0.8560	3.0040
Stove	2.85	1.310	1.14	0.288	0.0812	2.8192
Chestnut	....	2.370	1.90	0.556	0.0881	4.9141
Pea	....	....	0.866	0.333	0.1050	1.324
Buckwheat	....	....	....	....	1.8730	1.873

In general, this type of plant combines most of the qualifications of an ideal plant; its main disadvantages are: 1, the large individual units, with consequent tying-up of capacity when but a small amount of coal of a particular size or kind is to be stored; 2, expensive operation in the case of frozen coal, with liability to this difficulty from the method of making the piles. The coal can be handled with hot water if a supply is available, but this requires extensive drainage. This type is suited either to very extensive storage of hundreds of thousands of tons, or for the storage of moderate quantities of a single size, as for large steam-plants.

Suitable locations for plants of this type, while not common, are to be found; the most desirable is land with an average slope of about 1.25 per cent., not less than 600 ft. wide, for units on both sides of the central tracks, and at least a mile long. Enough space should always be left, and the tracks should be planned for extensions, which can readily be made by erecting additional units.

(*j*) *The Ransom Storage-System.*—A notable variation from the Dodge type was built under my supervision for the Lehigh Valley Coal Co., at Ransom, Pa., with a view to obtaining a plant at relatively low first-cost, for handling Western shipments. A place on the main line of the railroad, beyond the anthracite region, was selected.

The type of plant erected, Figs. 22 and 23, varies from the standard Dodge type in the use of a traveling trimmer-truss, building a wedge-shaped pile of coal with rounded ends, and reloading by conveyors in tunnels, with the assistance of traversing-reloaders, to a central loading-tower and screen-house. In detail, the coal is brought in on a double-track trestle, Fig. 24, with continuous bin-chutes controlled by gates similar to the trestle described in connection with the Hudsondale plant.

The cars are handled on this trestle by a rope-haulage system, spotted as desired and dumped into the hopper-chutes, from which the coal, to the capacity of the conveyors, is fed to a traveling-trimmer. This trimmer consists of a regular Dodge truss of 200-ft. span, the lower end resting on an eight-wheeled truck moving on a depressed track, and the other hung from a truck-frame on an elevated single-track structure, 83 ft. above the storage-floor.

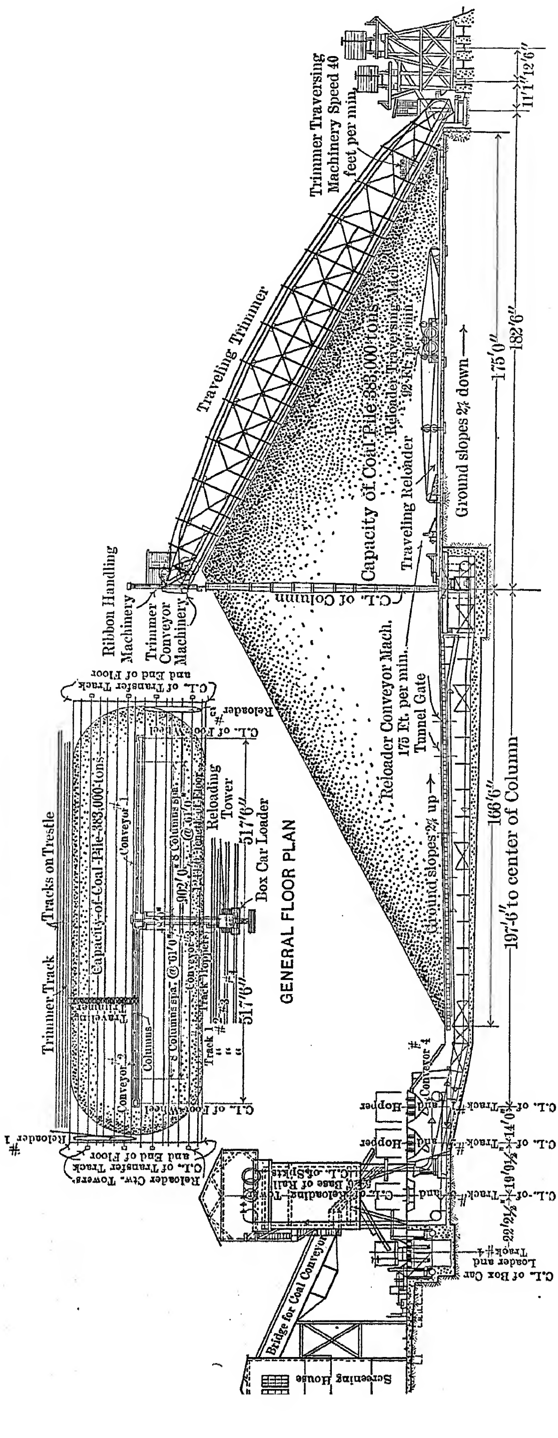

FIG. 22.—RANSOM PLANT, LEHIGH VALLEY COAL CO. PLAN AND CROSS-SECTION.

The track-support consists of steel trusses supported on 30-in. diameter cylindrical steel columns, built of 0.5-in. boiler-plate, supported at the bottom by ball-and-socket joints of large dimensions to avoid strains due to their resistance as a beam fixed at one end; the two central columns have longitudinal sway-bracing, and the colonnade as a whole is stayed from each column by side guy-ropes to anchorages outside the plant, and at the ends in a similar manner. All expansion of the colonnade, 902 ft. in total length, is transferred outward from the center, and expansion and contraction of the truss is taken care of by the hanging-support of its upper end. The upper and lower trucks are driven through their wheels by a 50-h-p. street-car motor, connected by shafting to both, and driving the trimmer 40 ft. per minute through suitable clutches controlled from the operating-house on the trimmer.

The trimmer-conveyor has flights 27 by 21 in., and is driven by a 125-h-p. motor, situated at the top of the truss and controlled from the operator's house. This conveyor is provided with the movable bottom previously described, and fed from the trestle-gates through an extensible chute, which is moved out to make a close connection.

Storage to maximum capacity is in the form of a single pile 1,240 ft. long, 342 ft. wide at the bottom and 83 ft. high, holding 380,000 tons of coal.

The plant will, at the most, handle only four sizes at once. These sizes are started in separate piles, and, if necessary, the coal is allowed to mix at the junction, a procedure admissible by reason of the exceptional rescreening facilities provided.

In designing the columns it was necessary to determine the probable strains from the reloading of the coal, for which no reliable data were available. An investigation disclosed the fact that in the original Dodge plants the timber masts in the coal failed by shearing from the avalanching of frozen coal, and not by bending, so the columns were given a factor of safety of six on the shearing-strength of successful masts, and further, it was planned to strengthen them by filling with concrete. This was attempted, but resulted in imperfect drying-out in the long closed columns, and damage to a couple of them from expansion due to freezing.

The trimmer-conveyors were proportioned on experience

with standard Dodge plants for a capacity of 180 tons per hour, but it was found that with the regular feed to capacity, made possible by the use of the pocket-chutes, and by moving the conveyor in case of delays in spotting or emptying cars, the actual work reached 3,800 tons in 10 hr., an object-lesson on the extent of the delays incident to the usual methods.

Reloading is accomplished by conveyors in longitudinal tunnels under the central colonnade, Figs. 22 and 25, delivering into cross bucket-conveyors at the center of the plant, which take the coal beyond the edge of the pile, and, turning, elevate it to the top of the screen-house. The longitudinal tunnels aggregate 1,035 ft. long, 8 ft. wide, and average 6 ft. 7 in. high, of reinforced concrete, with I-beam top; to permit of loading coal from the edge of the retreating pile without drawing under pressure, the top is in the form of a hoppered slot covered by short lengths of plank resting on the I-beams; one or more of these planks at the edge of the pile are removed to permit the coal to run into the tunnel-conveyors; these have top troughs close to the bottoms of the I-beams, guarded on the sides by steel aprons to prevent running over into the tunnel. The two conveyors, each with 10- by 30-in. flights, draw towards the center, where the coal slides gently into a cross-conveyor, with 24- by 38-in. buckets, which act as scrapers on the level, and, turning, form an elevator to raise the coal to the top of the loading-tower. At 20-ft. intervals through the tunnels, steel gates are provided to draw from any part of the pile in emergency.

The longitudinal tunnels also serve for foundations for the colonnade supporting the elevated center-trimmer track. Under the center of the plant the transfer from the longitudinal to the cross-conveyors is made in a concrete pit, with steel-and-concrete cover, averaging 24 ft. wide by 36 ft. long, which serves as an engine-room for the longitudinal conveyor-motors and driving-machinery.

A little more than half the coal in the plant is tributary by gravity to the central and cross tunnels, the balance is delivered into the longitudinal tunnel-conveyors by two traversing-reloaders, Fig. 25, similar in type to the standard pivoted Dodge reloaders. Each of these is 163 ft. long, of steel-truss construction, 13 ft. wide in the center, and carries an encircling conveyor

with 8- by 18-in. flights. The operating-machinery for each of these is carried in a house in the center. Current is supplied to the motors through flexible cables from plugs in the center tunnel. Traversing is accomplished through steel cables, two for each side of the plant, each passing around two 6-groove sheaves, and extending from end to end of the plant. Even with the six turns around the sheaves it was early found necessary to supply tension-towers at the ends of the plant to insure tractive power.

The operation of reloading is accomplished by drawing the center of the pile by gravity into the tunnel-conveyors, and following up with the traversing-reloaders to remove the two side-piles.

The central screen-house, Fig. 26, is amply provided with standard colliery shaking-screens, on which the standard mesh for the particular size of coal in course of reloading is placed. The screenings go across the track to a preparation-house, where they are separated into sizes and go to pockets for shipment or to the boiler-house for fuel. A feature of the screen-house is a transfer, similar to the cross-conveyor, which is arranged to take coal from open cars for transfer into box-cars, often preferred for Western shipment.

Rapid loading in box-cars is accomplished by the use of a Smith box-car reloader, a massive machine, Fig. 27, consisting of a platform resting on a cradle in the form of an arc of a circle, oscillating on supporting wheels, and provided with hydraulic mechanism for operation. When the box-car is in position for loading, and locked by power-operated clamps, the center of oscillation is near the top of the center door-opening, previously bulkheaded to the height of the top of the proposed loading. A 3-ft.-wide chute from the screen-house is extended into the car, and loading is commenced; as the car fills the cradle is gently revolved, tilting the car until one end is filled to the desired level, when the car is tilted in the opposite direction and the other end filled in a similar manner. It would seem, on first thought, that the coal already in would shift to the opposite end, as the car is reversed; but advantage is taken of the difference between the angles required for starting and for maintaining motion in coal, and the other end of the car is filled by the coal moving from the chute without shifting of the load.

The whole operation of spotting, clamping, loading, and releasing a 60,000-lb. capacity box-car can readily be performed in 6 min. under ordinary working-conditions, as compared with from 20 to 30 min. for loading with hand trimming.

Open cars may be loaded on this machine, or by chutes from the reloading-tower, under and beside which are shipping-tracks in addition to the one in front occupied by the reloader.

All the machinery is electrically operated by current supplied from a power-plant situated close to the reloading-tower, which is supplied with the finest portion of the screenings for fuel.

Extensive railroad-yards for both receiving and shipping are a portion of the plant, with ample trackage through the plant itself, both for storing and reloading.

The cost of the plant complete, including machinery, power-equipment, grading, tracks, reloading and transfer-tower, screen-house, dam, and a 0.5-mile pipe-line for water-supply, trestles, rope-haulage, and lighting, was very close to $1.15 per ton of capacity, and the operating-expense, excluding interest, taxes, and depreciation, is reported as low as 1.75 cents per ton handled during months of active operation.

No reliable data from a full clean-up are available as to breakage, but this appears to be somewhat greater than in a standard Dodge plant.

The plant as a whole has the advantages of low first-cost, cheap handling, large storage for the area occupied, ease and cheapness of extension, exceptionally thorough rescreening and ease of preparation of the screenings, low repairs, moderate maintenance, and very rapid handling. The disadvantages are inherent to the type: impossibility of handling more than one size at a time, in either stocking or reloading; partial mixing of sizes, except at a great sacrifice of capacity; limitation of number of sizes to not exceeding four; some fire-danger; and high depreciation on the wooden trestle.

(*k*) *Covered Storage-Plants.*—The difficulties from frozen and snow-covered coal, which are annoying in the latitude of New York, become so serious in more northern regions as to warrant expensive arrangements for their avoidance. As mere cold involves no difficulty in reloading, trouble from freezing is cured by the use of covered plants.

These comprise very many yards for wholesale and retail trade, usually of the trestle- or bin-type, hardly of a capacity to be dignified as storage-plants, and a number of plants along the Great Lakes of the bin-and-tunnel type, but except for being covered none of these vary materially from their general types as described.

A few covered plants in the mechanically-operated class vary so far from usual practice as to merit brief description.

The Hammond, Ind., plant of the Erie R. R., Fig. 28, of 60,000 tons capacity, a building 840 ft. long by 90 ft. wide, stores coal by a conveyor-system, with cross-conveyor in the roof. The sizes are separated by A-partitions and the walls sustained by anchor-bands in the coal itself. Reloading is accomplished by running the forward coal by gravity into a

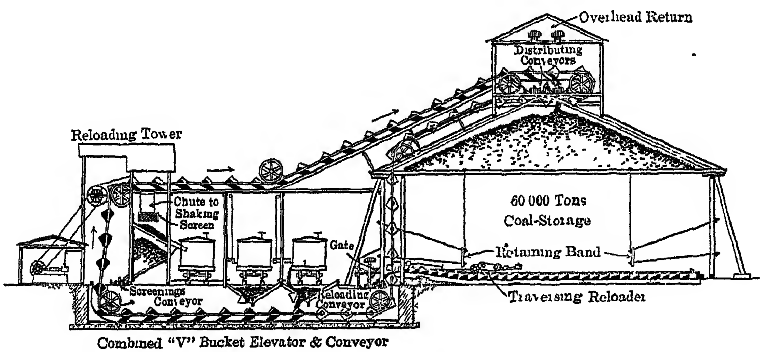

FIG. 28.—ERIE RAILROAD COVERED STORAGE- AND TRANSFER-PLANT, HAMMOND, IND. CROSS-SECTION.

longitudinal conveyor in front of the building, whence it is transferred to the return-buckets of the storing-conveyor, elevated to the loading-tower, screened and shipped. The screenings are prepared in a separate building. The balance of the coal in each pocket is delivered to the front conveyor by traversing Dodge reloaders, one serving each two bins. These are sheltered under the A-partitions when the bins are full.

This plant, which also is used as a transfer-plant, has the advantage of covered storage, moderate cost under the conditions, good handling-capacity and rescreening, with, as its most serious objections, fire-risk and excessive breakage from transfers between conveyors, and drop from the roof of the building in storing coal.

A better type, also designed by the Dodge Co., and erected for the Lehigh Valley Coal Co. at West Superior, Wis., to store coal from lake vessels, is practically a 50,000-ton trimmer-truss inclosed in a circular dome-shaped building, Fig. 29, The roof is supported by steel-dome construction and the low vertical sides by retaining-bands buried in the coal. Storing is accomplished by the use of the usual trimmer-conveyor with movable bottom, the only drop being for the first coal deposited until this makes a pile reaching to the point of trimmer entrance into the building. Reloading is accomplished by the use of a tunnel-conveyor extending to the center of the building, into which the coal tributary by gravity is admitted by valves in the roof of the tunnel. When all the coal thus available has been removed, a reloader, pivoted at the center of the building, has been uncovered and this delivers the balance of

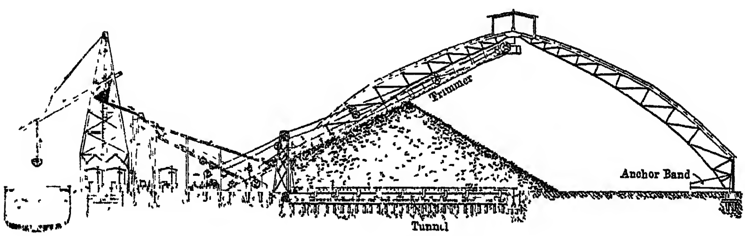

FIG. 29.—COVERED STORAGE-PLANT, LEHIGH VALLEY COAL CO., WEST SUPERIOR, WIS. CROSS-SECTION.

the contents to the tunnel-conveyor. All the coal is elevated by this to a loading-tower, where rescreening can be properly accomplished.

The cost of this plant, which comprises two such buildings, was about $3 per ton of capacity. Except for the breakage in unloading vessels, the stocking-breakage should but little exceed that of a standard Dodge plant, while the reloading-breakage would be somewhat greater by reason of the drop into the tunnel-conveyor, the necessity of drawing the first of the coal under pressure, and the double handling by reloader and tunnel of part of the coal.

The plant, being all of metal, is practically fire-proof, the main disadvantage being the lack of flexibility. Only one size of coal can, of course, be stored in each building, and any size

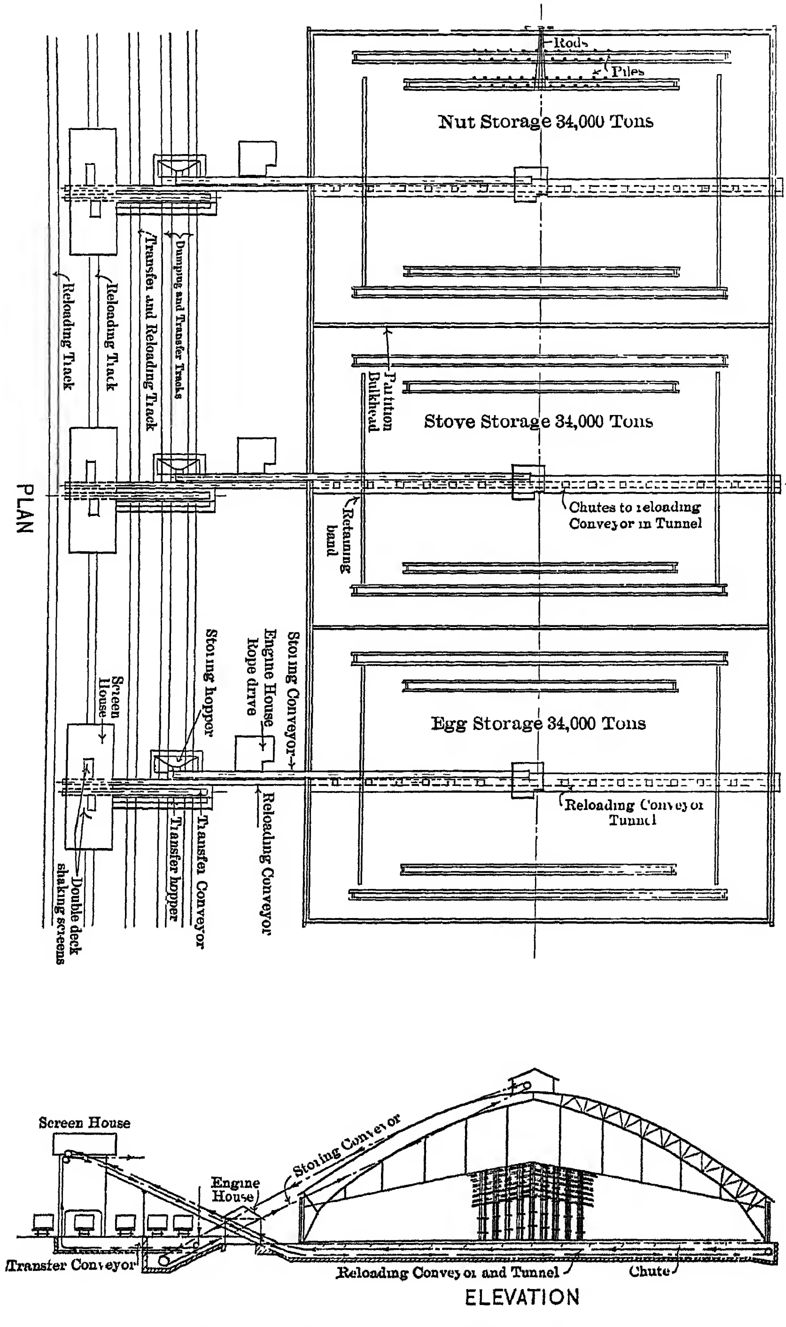

FIG. 30.—WENDE STORAGE-PLANT, LEHIGH VALLEY COAL CO., BUFFALO, N. Y. PLAN AND ELEVATION.

stored must be entirely reloaded before the building is available for a different size.

A covered plant of 100,000 tons capacity, built at Wende, near Buffalo, by the Lehigh Valley Coal Co., in 1906, Fig. 30, has also some unique features. The building is 480 ft. long by 250 t. wide. The front and rear walls, 20 ft. high, are braced by a retaining-band, and the end walls and two partitions are secured by tie-rods from double lines of piles. The curved roof is supported by steel trusses, the lower members of which are on the angle of repose of piled coal.

Each of the three pockets is provided with a central trimmer-conveyor for stocking, and a central tunnel-conveyor with valves on 14-ft. centers for reloading. The tunnel-conveyors carry the coal each to its own reloading-tower provided with proper screening facilities, and the coal which is not tributary by gravity to the tunnels is brought to them by dock-scrapers.

The driving is done by rope from a centrally-located engine. The cost of the plant approximated $2.25 per ton of capacity, and the operating-expense is said to be moderate. Breakage should approximate that of the plant previously described, over which this plant appears to have the advantages of lower first-cost, greater handling-capacity, less area occupied, and provision for three sizes of coal.

In addition to the plants described there are many others, particularly on the Great Lakes, showing interesting variations from their primitive types, but usually these modifications are on the general lines discussed.

VI. Extent of Storage.

The extent of storage installed by the various anthracite interests up to 1908, not including yard- or pier-storage at seaboard or lake points, amounts to 5,590,000 tons, as detailed in Table I.

TABLE I.—*Coal-Storage of Anthracite Interests.*

Type of Plant.

Owner.	Bin and Tunnel.	Hillside.	Tramway	Dodge.	Special and Covered.	Total.
	Tons.	Tons	Tons	Tons.	Tons	Tons.
Lehigh Valley R. R. and L. V. C Co.	300,000	225,000	150,000	310,000	600,000	1,585,000
Philadelphia & Reading	......	450,000	......	660,000	......	1,110,000
Pennsylvania R. R. and Susquehanna Coal Co.	......	......	......	880,000	......	880,000
Erie R R and Pennsylvania Coal Co.	......	120,000	......	385,000	60,000	565,000
Central R. R. of N. J., and Lehigh & Wilkes-Barre C. Co.	......	......	......	560,000	......	560,000
Delaware & Hudson	......	......	......	270,000	......	270,000
Lehigh Coal & Nav. Co.	......	......	......	240,000	......	240,000
Del., Lack. & Western R. R.	......	......	......	200,000	......	200,000
New York, O & Western	......	......	......	180,000	......	180,000
Total	300,000	795,000	150,000	3,685,000	660,000	5,590,000

VII. CONCLUSIONS.

In general, it appears that mechanical storage has distinct advantages over non-mechanical, that the Dodge type with its modifications is best suited to extensive storage-plants, and the traveling-tramway to smaller plants and to secondary wholesalers' installations.

All the non-mechanical plants involve such serious breakage in stocking as to warrant the greater first-cost of the mechanical types.

It is hoped that this review of practice in storing anthracite may lead to an appreciation of the controlling feature, the breakage of coal, which does not seem to be appreciated as

thoroughly as it should be, especially in the smaller plants and yards of the country, where better methods would be of distinct financial advantage.

That better descriptions of certain types and fuller data in regard to them are given, must be charged to my experience comprising plants of the trestle-and-tunnel, Dodge, hillside, Ransom, and traveling-tramway types, one or more plants of each of which types, with an aggregate capacity of 1,145,000 tons, have been constructed from my plans and under my immediate supervision, while descriptions of other types and variations are from inspection and study only.

Anthracite-Culm Briquettes.

BY CHARLES DORRANCE, JR., LANSFORD, PA.

(Wilkes-Barre Meeting, June, 1911.)

Introduction.

Culm is a general term used in the anthracite regions for many years to denote a mixture of coal, bony coal and impurities which is sent to the refuse-banks. Thus, 35 years ago culm contained the pea and buckwheat sizes of anthracite; but to-day, and as mentioned in this paper, culm is used specifically to denote the material which passes through the smallest screen in the anthracite-breaker. The smallest size of commercial anthracite is known as No. 3 buckwheat, barley, or bird's-eye coal, and is ordinarily made through a round-punched plate having openings $\frac{3}{16}$ in. in diameter, and over a round-punched plate with openings $\frac{3}{32}$ in. or $\frac{1}{16}$ in. in diameter. Thus culm will consist of coal, bony coal, slate, gravel, iron pyrite, etc., ranging in size from $\frac{3}{32}$ in. down to dust. Other local terms for culm are "slush," "silt," and "dirt."

The first experiments towards the utilization of anthracite culm by briquetting, and the first briquetting-work done in this country, were in 1872 at Port Richmond Piers, Philadelphia, Pa., by E. F. Loiseau.[1] Clay was used as a binder and the finished briquette was water-proofed with shellac, etc. Excessive cost was given as the reason for discontinuing work at this plant.

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[1] *Trans.*, vi., 214 (1877–78); viii., 314 (1879–80).

The Delaware & Hudson Co., in 1876, built a plant at Rondout, N. Y., which operated until 1880. Gas-tar was used as a binder, and anthracite screenings were briquetted for engine fuel. Excessive cost, poor results in firing, and the tendency of the fuel to cut the boiler-flues were given as the reasons for discontinuing the manufacture.

The next plant was built by E. F. Loiseau about 1878, at Nesquehoning, Pa., near the No. 1 or Nesquehoning colliery of the Lehigh Coal & Navigation Co. Pitch was used as a binder, and several samples of these briquettes were recently found along the bottom of the culm-heap at Nesquehoning. These briquettes are about 4 in. square, and contain coal from pea-size to dust. Except for rough corners, they are perfect in shape and have suffered little or no deterioration. The high cost of production was the chief reason for abandoning the work.

In 1890, a plant of English design was built at Mahanoy City, Pa., to make briquettes from anthracite culm for engine use on the Philadelphia & Reading railway. At first 18-lb. briquettes were made, but afterwards the size was changed to 2-lb. briquettes. The binder used was coal-tar pitch imported from England. The chief reasons for abandonment were inability to get a steady, uniform supply of binder, small margin between cost of manufacture and cost of coal, and poor results in practical use.

In 1905, the New Jersey Briquetting Co. erected a small plant in Brooklyn, N. Y., which was later moved to Perth Amboy, N. J., and is now in operation. The anthracite culm briquetted is shipped from the mines of the Susquehanna Coal Co. in the Lykens district. Melted coal-tar is used as a binder. The briquettes, of "pin-cushion" type, averaging in weight about 2 oz. each, with specific gravity of about 1.25, are used for domestic purposes.

In 1906, the Scranton Anthracite Briquette Co. erected a plant at Scranton, Pa., near the Storrs colliery and washery of the Delaware, Lackawanna & Western railroad. The plant consists of one press of the roll, or Belgian, type, said to have a capacity of 500 tons per day of 10 hr. The Delaware, Lackawanna & Western railroad uses 200 tons of these briquettes daily on freight-locomotives, burning them mixed with No. 1 buckwheat and bituminous coal.

In 1908, the Lehigh Coal & Navigation Co. built a small experimental plant at Lansford, Pa., and in 1909 began marketing briquettes from this plant for household use. The plant was destroyed by fire in December, 1909, and a new plant was put into operation in March, 1911, having two presses of the Belgian type, and a capacity of from 15 to 20 tons per hour.

This practically reviews to date the history of the briquetting of anthracite culm in the United States. Many small ventures have been started and companies formed, some of *bona fide* producers, but the greater number being stock-selling propositions, or secret-binder exploitations. It will be noted that there has been but one of the anthracite producing companies—the Lehigh Coal & Navigation Co.—which has directly done any work along these lines since the experiments in 1876 of the Delaware & Hudson Canal Co. The failures of the first four plants naturally had a great deal to do with this condition, but when we consider that, due to cheaper pitch and better machinery, the cost of briquetting has probably been materially reduced since 1890, the date of the abandonment of the last plant, while the cost of anthracite coal has steadily increased and will probably continue to increase, the two principal reasons for these first failures seem to be to a great extent eliminated.

E. W. Parker, in a publication of the U. S. Geological Survey, gives as a reason for the inactive attitude of the anthracite companies in regard to briquetting that it would tend to reduce the output of anthracite proper, and that this would mean an increase in the cost of production due to fixed charges. Another reason has been advanced that the anthracite companies would be "competing with themselves," if they started in to produce and sell briquettes. Both of these arguments are answered by the fact that every mining-man in the anthracite regions is working tooth and nail to get the greatest number of tons of prepared-size coal per mine-car, and if, at a fair profit, he can convert the worse than worthless culm into a prepared-size fuel, he accomplishes the same result in the end. The large briquetting-plants of Europe are operated and controlled by the coal-producing companies, and it seems reasonable to expect that any large development of the briquetting of anthracite culm in the United States must depend upon the support of the anthracite producing companies.

The commercial tonnage of anthracite per year is about 60,000,000 tons, and a conservative estimate of the amount of culm produced annually would be about 8 per cent. of the commercial tonnage of coal, or about 5,000,000 tons. This percentage is higher in the Southern and lower in the Northern anthracite-field. The average cost of disposal of this culm is about 2 or 3 cents per ton, depending on the methods employed and the local conditions. In many cases the culm is flushed back into worked-out rooms or chambers in the mines, and by this method the intervening pillars removed. This disposal is often given as an argument against briquetting, but it is hardly admissible, since our German brother finds that sand and gravel are better for the purpose, and he refuses to use good coal, for which he has paid to mine and prepare, as a substitute for non-combustible material. At all events, the method of using anthracite culm for mine-filling can hardly be classed as a step towards the scientific conservation of natural resources. In most cases, where culm is not flushed back, it is either mixed with jig- and platform-slate and sent to the refuse-bank, or washed into slush-dams and there settled. The former method means that the culm is practically thrown away for good, since it is doubtful whether it could be extracted from the banks except at great expense, unless worked again for washery purposes, and to-day few refuse-banks are rich enough for that treatment. The latter method is cheap, and the culm can be reclaimed easily at a small cost. One of the large anthracite companies has ordered that all its culm must be kept separate from other refuse, having in view its reclamation at some future time.

Leaving aside the value received from culm used for flushing-purposes, to-day the annual production of 5,000,000 tons of culm, averaging 75 per cent. of pure coal, entails, in addition to a cost per ton of mining equal to that of the best chestnut-coal, a further cost of 2 or 3 cents per ton for disposal, with a revenue which is negligible. As an economic question, therefore, its utilization is a most intensely interesting subject.

There are two general methods which have been suggested for using culm for fuel: one, combustion, either in pulverized state or without pulverizing, in special furnaces; the other, to briquette it. The first method will not be taken up in this paper, but from a question purely of fuel-value in dollars it is perhaps the better way, if feasible. But little expense, either in labor or

material, is added to the culm, if burned direct, and thus the resulting cost per British thermal unit, or per horse-power produced, is small. On the other hand, the efficiency of combustion will probably be low, and its use adapted only for steam-producing purposes, where the fuel must have a very low selling-price. Briquetting, on the other hand, necessitates considerable addition of both material and labor before the briquette can be made. The resulting fuel, however, can compete with the higher-priced domestic fuels, and has a high fuel-efficiency. From an academic stand-point briquetting also has the further advantage that if briquettes are used for domestic purposes it will mean a 15 per cent. longer life to anthracite as a domestic fuel, and advances the day when the cost of production of anthracite will bring its selling-price beyond the reach of all save the well-to-do. It is a question whether the difference between the cost of briquettes and their selling-price will not be as great as the value received for the culm itself for steam-raising purposes.

Experimental Work.

About the middle of 1907, the Lehigh Coal & Navigation Co. began experiments towards the utilization of culm. The investigation was in charge of George B. Damon, Fuel Engineer. A small laboratory was established in Mauch Chunk, Pa., which a few months later was transferred to the mines at Lansford. After preliminary investigation, it was decided to do the first work in briquetting and later to take up experiments in direct combustion.

The first work done was a thorough systematic series of tests to determine both the physical and chemical characteristics of the culm from the various collieries of the company. Tests were also run on culm from collieries of the other large anthracite companies. These tests were made as follows: A 10- to 15-lb. sample of culm was taken from settling-tank elevators every 5 min. during an entire breaker-day of 9 hr., giving, at the end of the day, a general sample of approximately 1,200 lb. This large sample was thoroughly mixed at the collieries and quartered down to from 80 to 100 lb., which was placed in sample-boxes and sent to the laboratory. At the laboratory the sample was quartered once, two alternate quarters being used for the determination of mesh-sizes, and the other two alternate quarters for analysis. The latter portion was carefully quartered

to a 5- to 10-lb. sample; dried, crushed, and quartered to an 8-oz. sample; which, in turn, was pulverized and quartered to four 2-oz. samples, and each analyzed for moisture and ash. The sample for the mesh-determination was quartered to from 10 to 15 lb., dried and thoroughly screened, the percentage of weight being taken of the following sizes:

1. Material over a No. 1 buckwheat mesh ($\frac{5}{16}$ in. round).
2. Material through No. 1 and over No. 2 buck. mesh ($\frac{3}{16}$ in. round).
3. Material through No. 2 and over No. 3 buck mesh ($\frac{3}{32}$ in round).
4. Material through No. 3 buck. and over No. 10 wire mesh.
5. Material through No. 10 mesh and over No. 20 wire mesh.
6. Material through No. 20 mesh and over No. 40 wire mesh.
7. Material through No. 40 mesh and over No. 60 wire mesh.
8. Material through No 60 wire mesh.

When the percentage by weight of these different mesh materials had been found, a separate analysis for ash and moisture was made on each size material. From these data the ash-content of the general unsized sample was calculated and checked against the mean analysis of the four determinations made on analysis sample. Determinations were also made—both mesh and chemical—on the fine material going through the No. 60 mesh on a number of samples,—namely, on the material staying on No. 80 mesh, No. 100 mesh, and material passing through the No. 100 mesh. The weight per cubic foot of the dry culm, and of the different mesh materials, was determined, both loose and packed. Moisture-determinations were also made on culm as received. The above procedure was continued over a period of about four months, until each colliery had been sampled for about 15 different days scattered over the above period.

The results of this work indicated the fallacy of the prevalent idea that all culm is largely pure coal. On the contrary, as a general rule, the finer material in culm was found to carry higher ash than the coarser part of the same sample. The only exception to this was that the material passing through the No. 100 mesh was found slightly lower in ash in all cases than the material staying on the No. 100 mesh, but the difference was very slight, being at the maximum 1 per cent. Typical examples of these results, obtained from two samples from the Southern and two from the Northern anthracite-field, are given in Table I.

TABLE I.—*Ash-Content in Sized Anthracite Culm.*

	Northern Anthracite-Field.				Southern Anthracite-Field.			
	Sample No. 1.		Sample No 2.		Sample No 1.		Sample No. 2.	
	Weight of Material.	Ash.	Weight of Material.	Ash	Weight of Material	Ash.	Weight of Material.	Ash.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent	Per Cent.
Over No. 1 buck. screen,	0.20	21.1	.. .	. .....	0 25	22 60	5.95	23.40
Over No. 2 buck. screen,	4.30	34.1	0 05	. .....	1.65	19.11	2 70	29.90
Over No. 3 buck. screen,	12.70	31.9	6.30	22 17	3 35	21.75	18.30	33.72
Over No 10 wire mesh, .	0 70	30 9	1 75	12.80	0.90	20 32	1 00	30 10
Over No 20 wire mesh, .	25 90	34.1	30.20	16.84	33.60	20 75	19.55	31.45
Over No 40 wire mesh, .	28.50	36.5	24.13	19.17	31.40	18.80	27.35	29.31
Over No. 60 wire mesh, .	20 21	39 5	17 87	21 88	20.90	21.16	16 40	34 95
Over No 80 wire mesh,	3 68	43 5	5.38	23.61	3 45	28.30	3 10	36.40
Over No. 100 wire mesh,	0.80	44.1	} 13 20	28.93	1 10	32 10	1 10	44.15
Through No. 100 wire mesh, ...	3.01	43.0			3.30	31 60	3.90	43 60
	100 00		98 88		99.90		99 35	
General sample,		37.5		18.80		20.60		30 56

Of the material passing through No. 100 mesh about 75 per cent. will stay on a No. 200 mesh. The weight per cubic foot of dry culm will average about 55 lb. The moisture from culm made in a wet breaker will vary from 30 per cent. when first loaded, down to from 10 to 15 per cent. after more or less drainage in cars.

After these tests had been finished and the results tabulated, the following conclusions were reached in regard to the briquetting of this material:

1. Size.—The material was readily adaptable without crushing, but it was judged that the very finest material (passing No. 60 mesh) would perhaps have a deleterious effect on the resulting briquette, and should be rejected.

2. Moisture.—The culm would probably have to be dried before briquetting.

3. Impurities.—It was decided that a method of reducing the impurities from the coal must be devised, if the briquette was to be used for a domestic fuel.

Tests indicate that culm averages about 25 per cent. incombustible material, which is probably much too high for a satisfactory domestic fuel. Due to wide variations, the ash in culm may run as high as 40 per cent., and the nature of the impurities is such as to cause trouble. The finer the material the

larger the amount of pyrite it carries, and the material through No. 60 mesh, when panned in a prospector's gold-pan, will show on the clean-up a thick covering of "fool's gold."

As drying at some period of the briquetting-process was deemed advisable, it was decided to investigate the feasibility of pneumatic separation, and an experimental separator was built of the design shown in Fig. 1. It consisted of a narrow box-like compartment about 13 ft. long, 18 in. wide, and 10 ft. high. In the front end at the top was placed a hopper, from which the culm was fed by gravity on to a feeding-belt, which

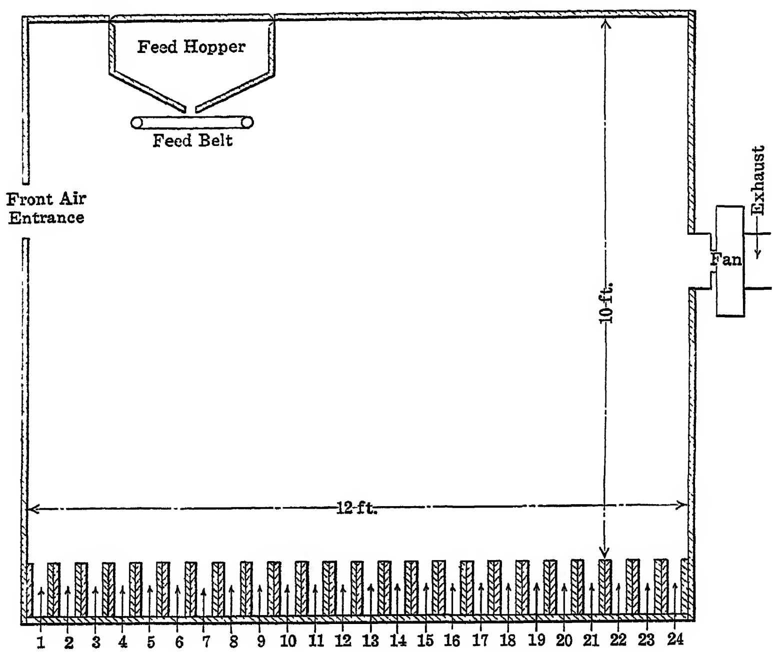

FIG. 1.—SECTION OF TESTING SEPARATOR.

in turn fed a steady shallow stream of culm into the separator at a point just above the opening in the front end of the separator. At the back end of the separator was another opening which led to a Sirocco multivane fan. The bottom of the separator consisted of 24 small boxes, or bins, each about 6 by 18 by 18 in. in size. These boxes fitted closely to each other, and air-tight doors were made along the bottom of the separator, so that after each test they could be removed. Both the front and rear openings of the separator were adjustable

in vertical position and in area, and the positions of the feed-hopper and belt were also adjustable. There were air-tight windows in the top and sides of the separator, and electric lights were placed in its interior, so that observations could be made during the tests. The method of operation was: The fan being started, a current of air was sucked past the front opening on through the box-like body of the separator, and exhausted by the fan outside the laboratory. A thin layer of dry culm was fed into this air-current at the front opening, and carried towards the rear of the separator. The impurities in the culm, having a specific gravity ranging from 2.5 for slate to 5.2 for pyrite as compared with 1.6 for coal, should settle out of the current first, while the lighter coal should be carried further along. The above action was found to take place to a certain degree. The tests were as follows: A weighed quantity of dry culm, previously analyzed for ash and mesh constituents, was placed in the feed-hopper, and the position of the hopper and the area of front and rear openings were noted. The velocity of air at the front and rear openings during each test was taken continuously by anemometers, and the speed of the fan, which was changeable by a variable-speed motor-control, was noted. The thickness of the layer of culm fed and the duration of the test also were noted. When the test was finished, the doors at the bottom of the separator were removed and the small boxes, or bins, taken out, their contents weighed, screened for the different mesh material deposited therein, and analyzed for incombustible. All these data for each test were tabulated on a regular printed form.

These tests were run for a period of about six months, until all variations of air, openings, feed, etc., had been tried. Figs. 2 and 3 show diagrammatically the results obtained. The abscissas are the small boxes or bins in the bottom of separators, while the ordinates are either percentage of incombustible, percentage by weight of culm, or percentage by weight of the different mesh material. The curves show in a general way the amount, sizes, and percentage of ash in the culm deposited in the boxes.

The condensed results of these tests showed that under the most favorable conditions, from 50 to 60 per cent. of the culm

could be obtained as briquetting-material from the separator, with a reduction in ash of from 2.5 to 3.5 per cent. This means that nearly half of the original culm would have to be thrown away, and the culm recovered would still be nearly as

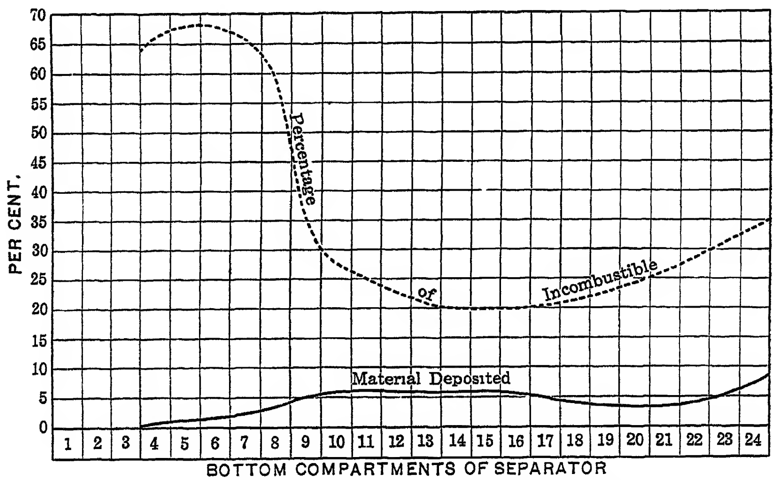

FIG. 2.—AVERAGE OF TESTS ON UNSIZED CULM.

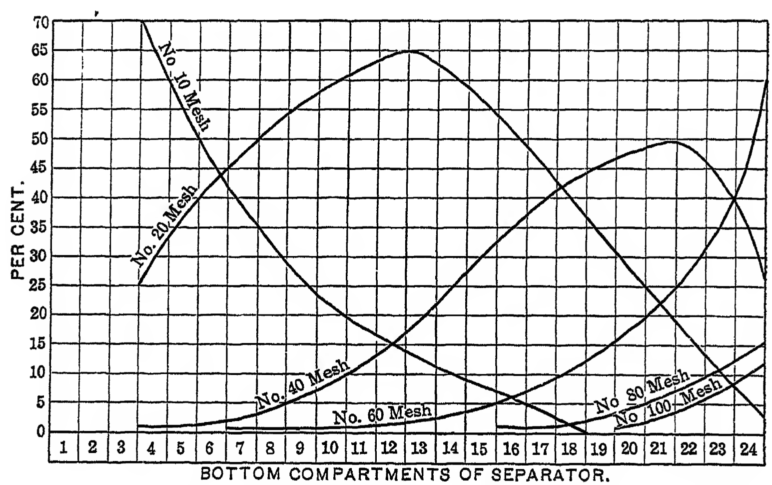

FIG. 3.—AVERAGE OF TESTS ON UNSIZED CULM. PERCENTAGE OF MESH MATERIALS DEPOSITED PER COMPARTMENT OF SEPARATOR.

high in ash as it was before treatment. A great deal of the worst impurities was eliminated, however, and the briquetting-material recovered was very much improved in size, since the oversize and the very fine dust were eliminated. The main

cause for these poor results was the fact that the culm was not sized closely enough for good separation. The small pieces of slate and the large pieces of coal, both having the same weight, would fall into the same box.

From the above tests, however, a process was outlined and patented, the patents being assigned to the Lehigh Coal & Navigation Co. These patents control both the machinery and the process. In a general way, the process as outlined is to dry the slush, and separate by air into three products from the separator: (1) Material which settles in the front end of the separator and is large in size. This is to be used in a gas-producer to generate gas, which in turn is to be used to dry the original culm before separation. (2) Material which settles out of the air-current in the middle of separator and is to be used as briquetting-material. (3) Very fine material, the last to settle at the rear end of separator, is to be burnt direct in a special furnace, on a surface which is preheated by producer-gas from the first product.

Table II. gives the average physical and chemical properties of these three products, as shown by results obtained in tests:

TABLE II.—*Products of Pneumatic Separation.*

	(1) Producer-Material. Per Cent.	(2) Briquette-Material. Per Cent.	(3) Furnace-Material. Per Cent.
Quantity of the original culm by weight,	25	55	20
Incombustible,	28	20	25
Quantity of material, 10-mesh or larger,	30	2	0
Quantity of material between 10- and 60-mesh,	68	90	35
Quantity of material smaller than 60-mesh,	2	8	65

Experimental Plant.

A small experimental plant to demonstrate the process was built in the fall of 1908. As shown in Fig. 4, the equipment consisted of a Bartlett & Snow rotary drier 3 ft. in diameter and 15 ft. long, to which the culm was fed from a track-hopper. The dry slush was elevated after leaving the drier to a Damon air-separator, in principle like the small testing-separator, Fig. 1, but with three bins at the bottom instead of the 24 small boxes. The front bin collected. the producer-material, which was elevated to the charging-platform of a 250-h-p. Wile suc-

tion gas-producer. The producer consisted of the producer proper, the scrubber, and the receiver, and the gas produced was burned in the furnace of the rotary drier. The middle bin of the separator collected the briquetting-material, which was conveyed in a screw-conveyor to the briquetting-press. The pitch binder, measured and pulverized, was fed into this conveyor with the coal, and the mixture was elevated to the mixing-tower of the briquetting-press, where superheated steam was introduced and the pitch melted. The heated mixture was then fed to the press. The briquetting-equipment was made in Belgium by Robert Devillers, and the press was the ordinary roll type, making egg-shaped briquettes, each weighing 1.5 oz. The third bin collected the fine furnace-material,

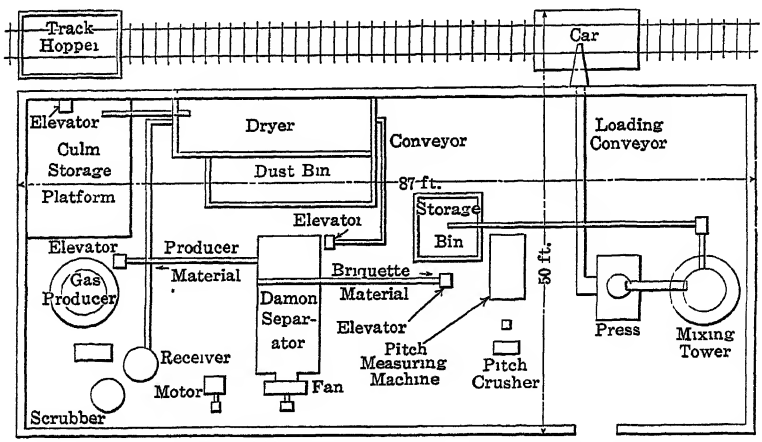

FIG. 4.—PLAN OF EXPERIMENTAL BRIQUETTING-PLANT.

which was rejected because no provision had been made for a furnace to burn it, due principally to lack of space.

The plant was started about the middle of November, 1908, and experiments were made during the winter of 1908–1909; but the work was not entirely successful. It was found that the producer-material was too fine to be used in the gas-producer, and No. 1 buckwheat had to be used in its place, the producer-material being rejected. The drier was not adapted for drying the slush, and its capacity was very small. However, about 500 tons of briquettes were manufactured during the winter, and burning-tests, etc., made on them. Various pitches were tested as binders, and a great deal of data obtained on binders,

power required, drying, and pressing. In the spring and summer of 1909, more than 1,000 barrels of briquettes were sent as samples to various retail coal-dealers in the Eastern States, and

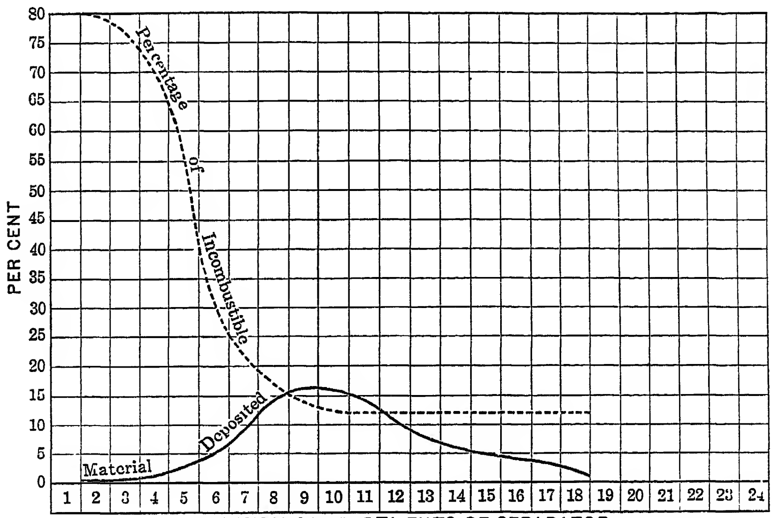

FIG. 5.—AVERAGE OF TESTS ON SIZED CULM OVER NO. 20 MESH.

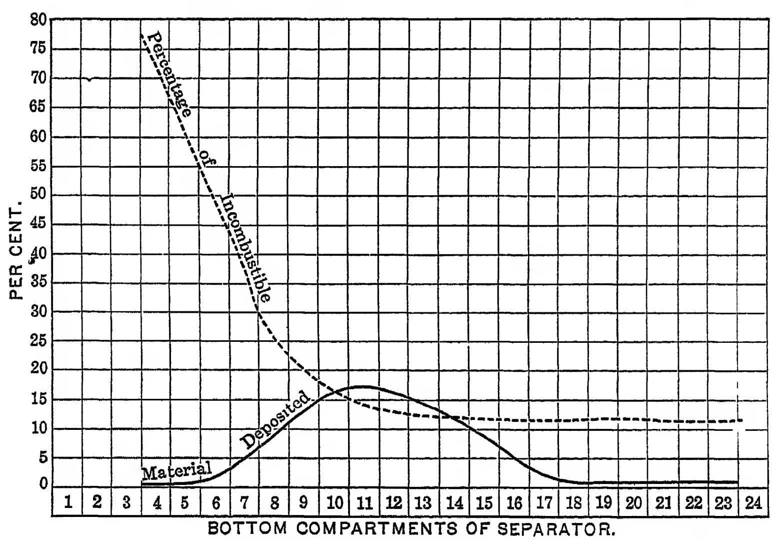

FIG. 6.—AVERAGE OF TESTS ON SIZED CULM OVER NO. 40 MESH.

the results were so favorable that in June the plant was started to manufacture boulets (as the smaller briquettes are called) for the domestic trade. Due to the fact that the capacity of the drier

was so small, the press could not be worked steadily, but the drier was operated 24 hr. per day, which gave enough material to make about 40 tons of product per day. The boulets found a ready sale in New England, and the plant was worked steadily up to December, 1909, when it was destroyed by fire. During this time about 6,000 tons of boulets were marketed, all for domestic consumption.

The results from a sales stand-point were so favorable that work was immediately begun on designs and plans for a commercial plant. Several different plans were discussed before reaching a final decision. Some time prior to the fire, tests had been made in the small testing-separator with sized culm, which showed that if the culm was screened into four sizes—material over No. 10 mesh, material over No. 20 mesh, material over No. 40 mesh, and material over No. 60 mesh—and each

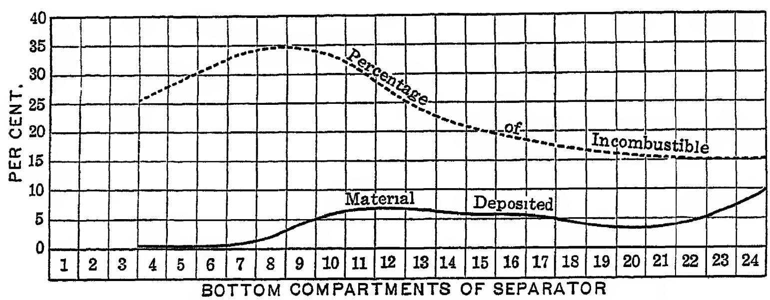

FIG. 7.—AVERAGE OF TESTS ON SIZED CULM OVER NO. 60 MESH.

mesh material run through the separator by itself, the results were very much better than those obtainable with the unsized culm. A recovery of 75 per cent. of the original culm was made, with a reduction in ash of from 6 to 8 per cent., which meant a combustible value of the resulting briquette as high as the average nut or stove anthracite. It was decided, therefore, to size the culm before separation in the new plant. Diagrammatic results of these tests on sized culm are shown in Figs. 5, 6, and 7.

The New Plant.

The construction of the new plant was commenced in the fall of 1910, and the plant put in operation in March, 1911. Fig. 8 gives a ground-plan, and Fig. 9 a general view of the plant.

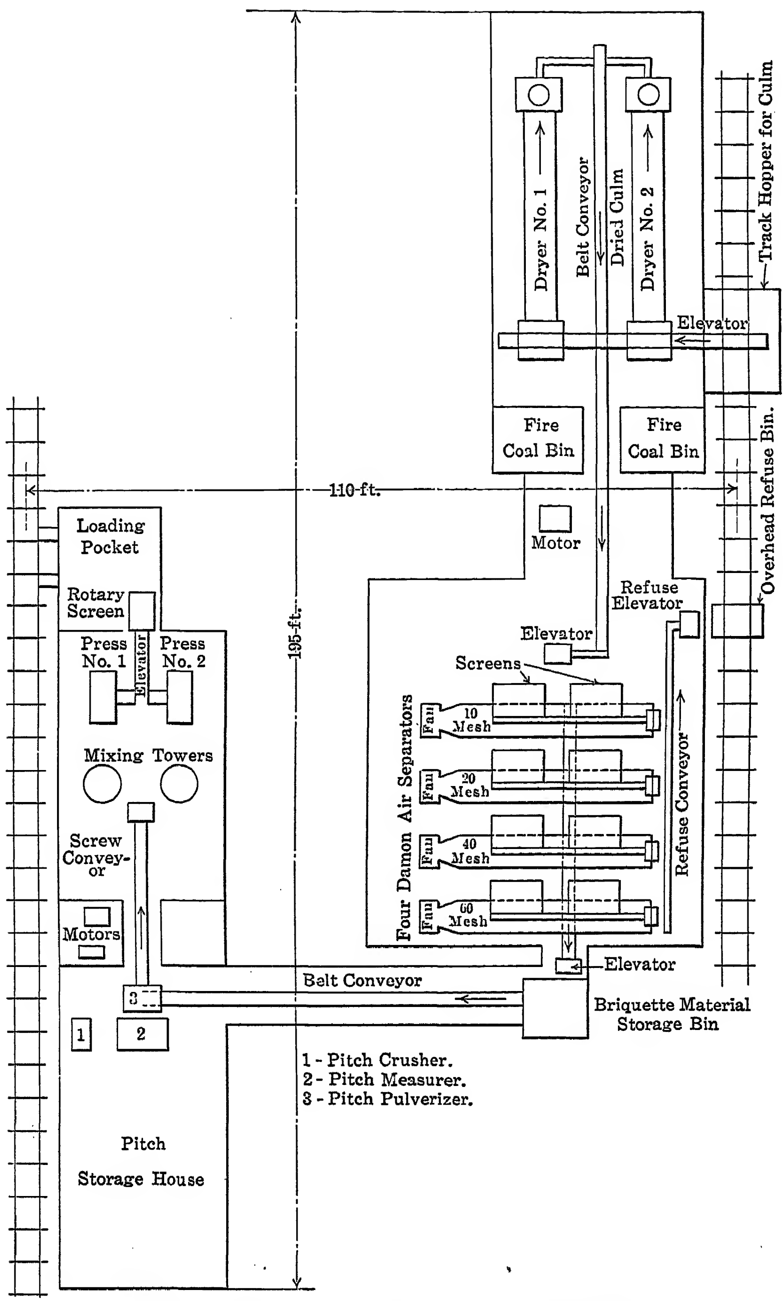

FIG. 8.—PLAN OF PRESENT BRIQUETTING-PLANT.

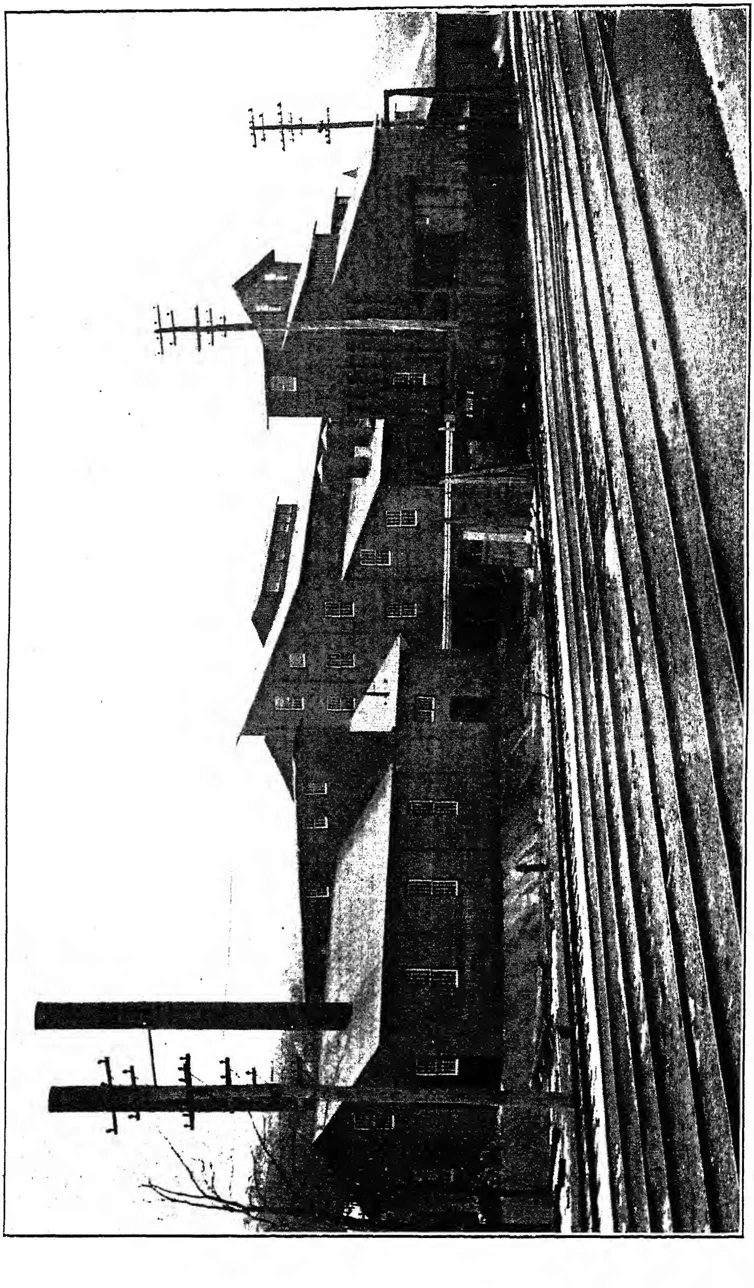

FIG. 9.—VIEW OF THE BRIQUETTING-PLANT FROM THE NORTHEAST.

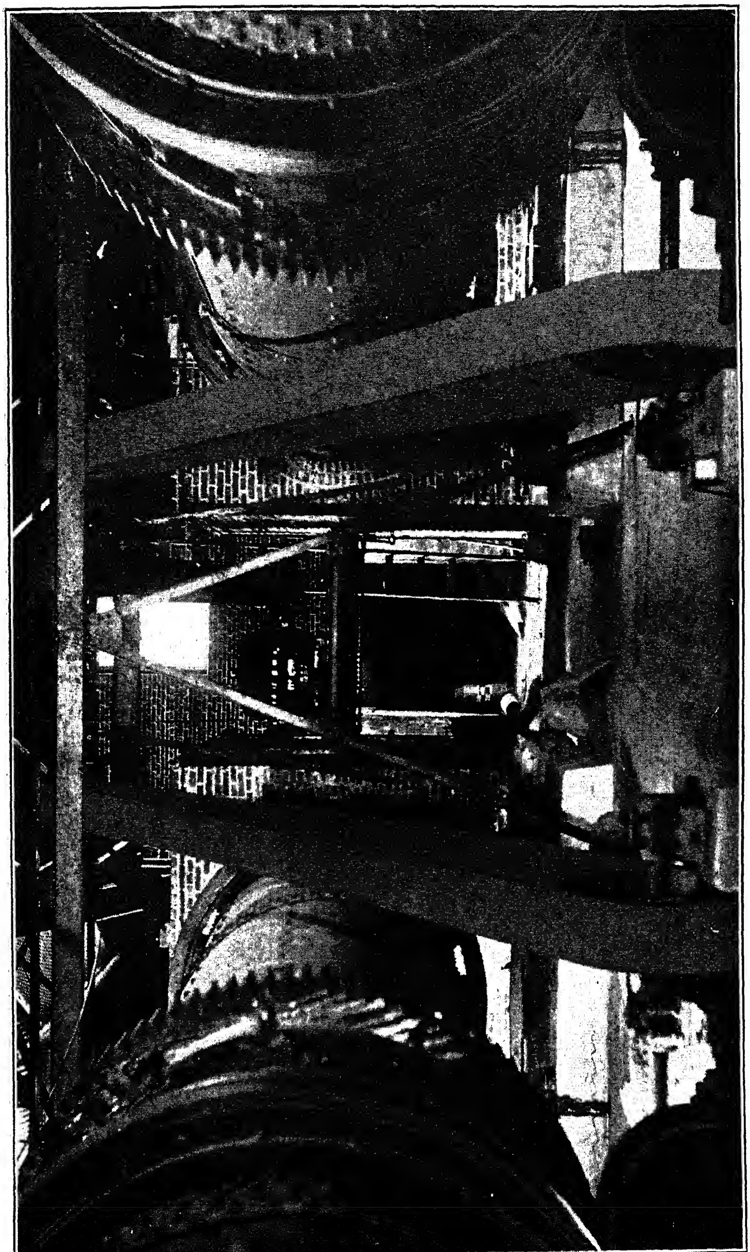

FIG. 10.—VIEW OF INTERIOR OF DRYING-BUILDING.

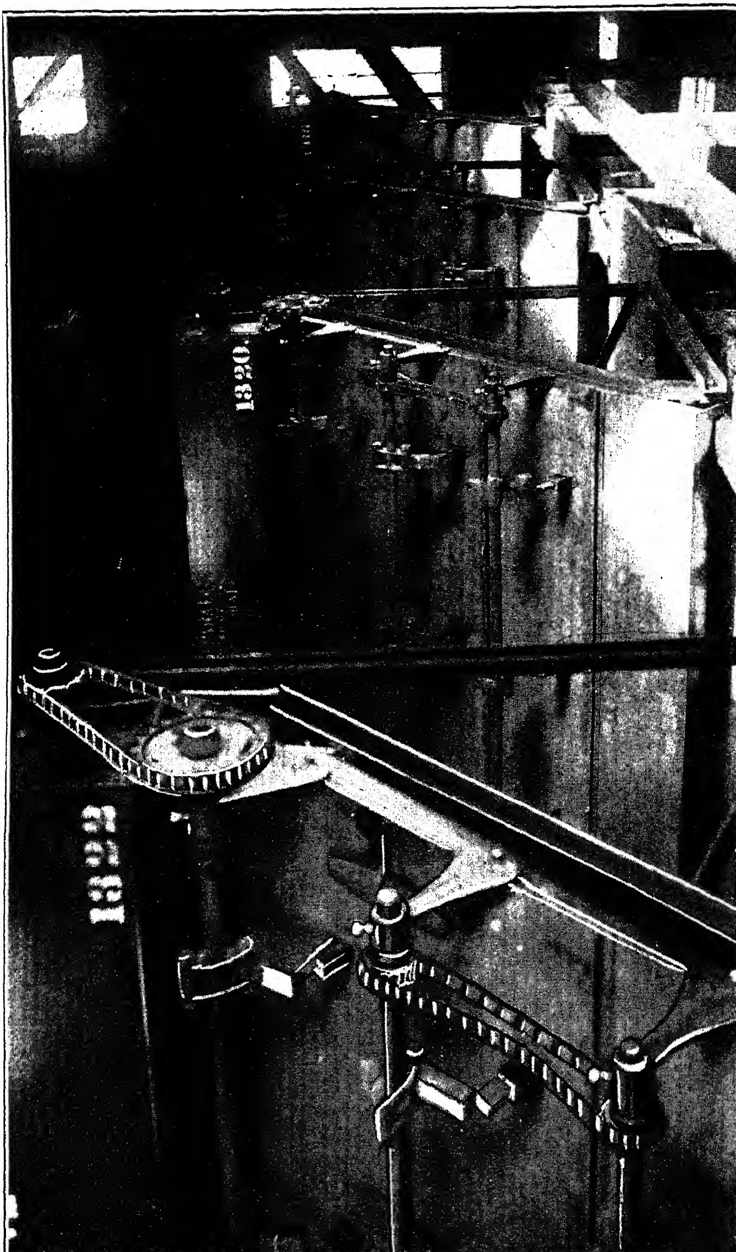

Fig. 11.—View of Screening-Floor.

The operation, as now conducted, is as follows: The culm is loaded into steel gondolas of 100,000-lb. capacity at the Lansford colliery of the Lehigh Coal & Navigation Co. and dropped to the siding of the briquetting-plant, which is situated close to this colliery. The culm is dumped into a track-hopper and elevated to the drying-plant. Here the culm passes through two 30- by 6-ft. Vulcan rotary-kiln driers, Fig. 10, and is dried by direct contact with the heated gases of the drier-furnace. The dried culm is then conveyed on a belt-conveyor back to the separating-building, where it is elevated to the top or screening-floor. Here the culm is run over four sets of Newago vibrating-screens, Fig. 11. The first set of screens has an extra scalping-screen arrangement by means of which any commercial sized coal which may be in the culm is saved and returned by chutes to the drier-building, where it is burned in the drier-furnace. The very fine culm passing through the last set of screens is conveyed to the refuse-conveyor.

Four Damon air-separators are placed under the four sets of screens, and the sized culm from each set of screens is fed to each separator. Fig. 12 is a section showing the feed and action of one of these separators. Fig. 13 illustrates the feed-device of the separator. The refuse or slate from each separator feeds into a screw-conveyor running along the fronts of separators, as shown in Fig. 14. This conveyor feeds into an elevator, which discharges the refuse into an overhead steel bin, from which it is discharged into railroad-cars and sent to the Summit Hill mine-fire for slushing. The purified culm from each separator feeds into a screw-conveyor running under the four separators, and is elevated to 50-ton storage-bin.

The culm is fed in a measured stream from the storage-bin on to a belt-conveyor, which takes it to the mixing-house, where the binder is added. Coal-tar pitch, used as a binder, is cracked to "pea and dust" size in a set of rolls. The cracked pitch is elevated to a pitch-measuring device, which, by means of a friction spool-and-wheel drive, has variable speed and feeds a measured amount of pitch to the squirrel-cage pulverizer, which pulverizes the cracked pitch, and feeds it into a screw-conveyor along with the measured stream of culm from the belt-conveyor. Fig. 15 shows this equipment.

The dry mixture of pitch and culm is conveyed to the bri-

quetting-building, and elevated to the two mixing-towers of the presses. Here the mixture is heated with superheated steam and the heated mixture is fed to the presses. The briquettes are elevated directly from the presses to the bin, from which later they are loaded into cars. Before dropping into the pocket the briquettes pass over a rotary screen; removing the fine material, which is returned to the press. Fig. 16 is a general view of one of the presses and the mixing-towers.

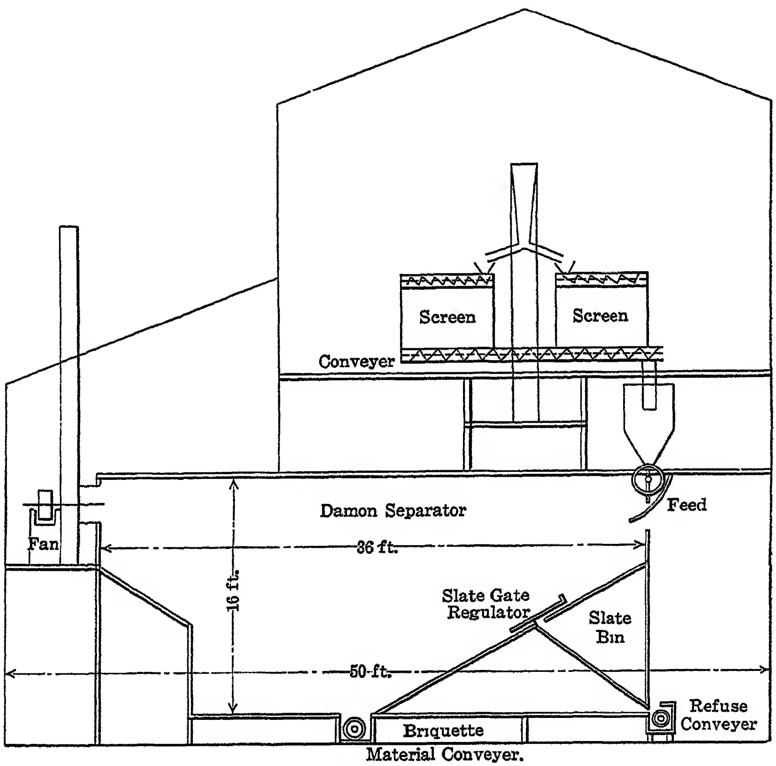

FIG. 12.—SECTION OF SEPARATING- AND SCREENING-BUILDING.

The operation of the plant and process to date has been successful. No large mechanical troubles have been encountered. Tests of power-consumption, drying, and separating have been conducted, and results have been up to expectation. The cost of manufacture during the first month of operation, with all the mechanical troubles consequent to starting up, and a production of less than one-seventh the full capacity, was within a few

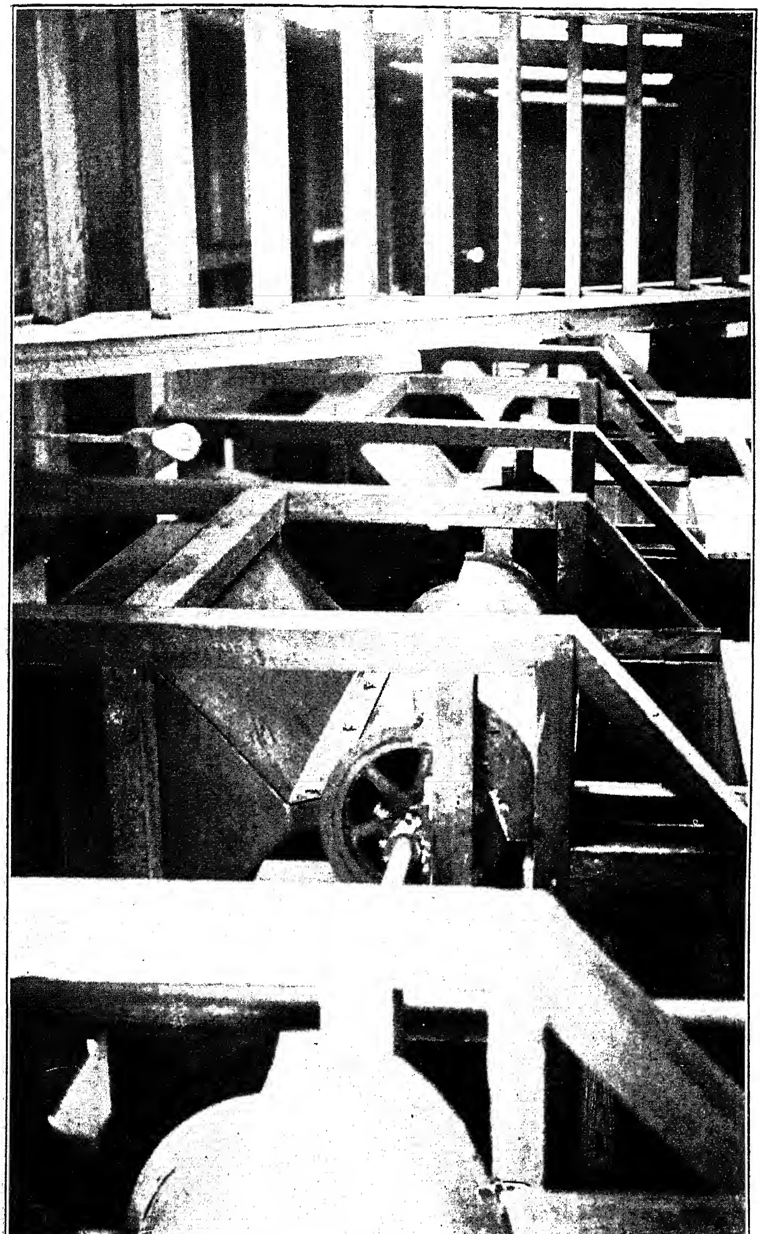

FIG. 13.—FEEDING-DEVICE ON DAMON SEPARATORS.

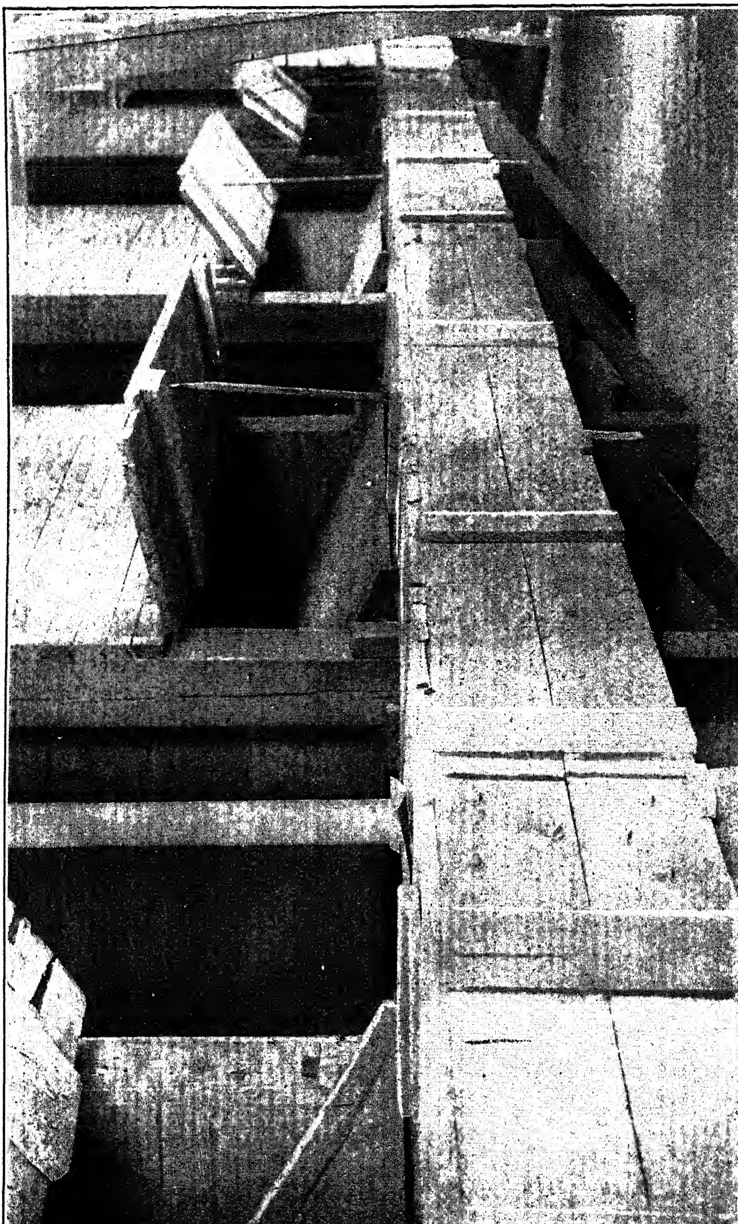

FIG. 14.—REFUSE-DISCHARGE, DAMON SEPARATORS.

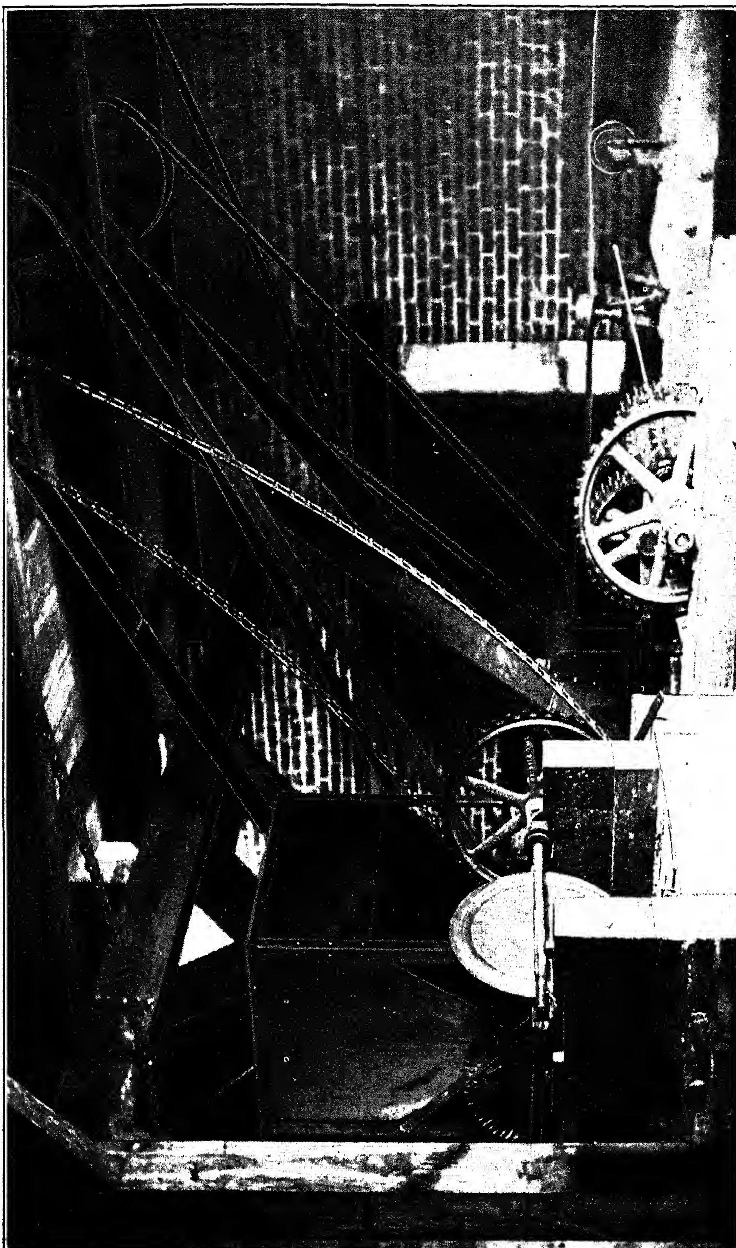

Fig. 15.—Pitch-Measuring and Pulverizing Machinery.

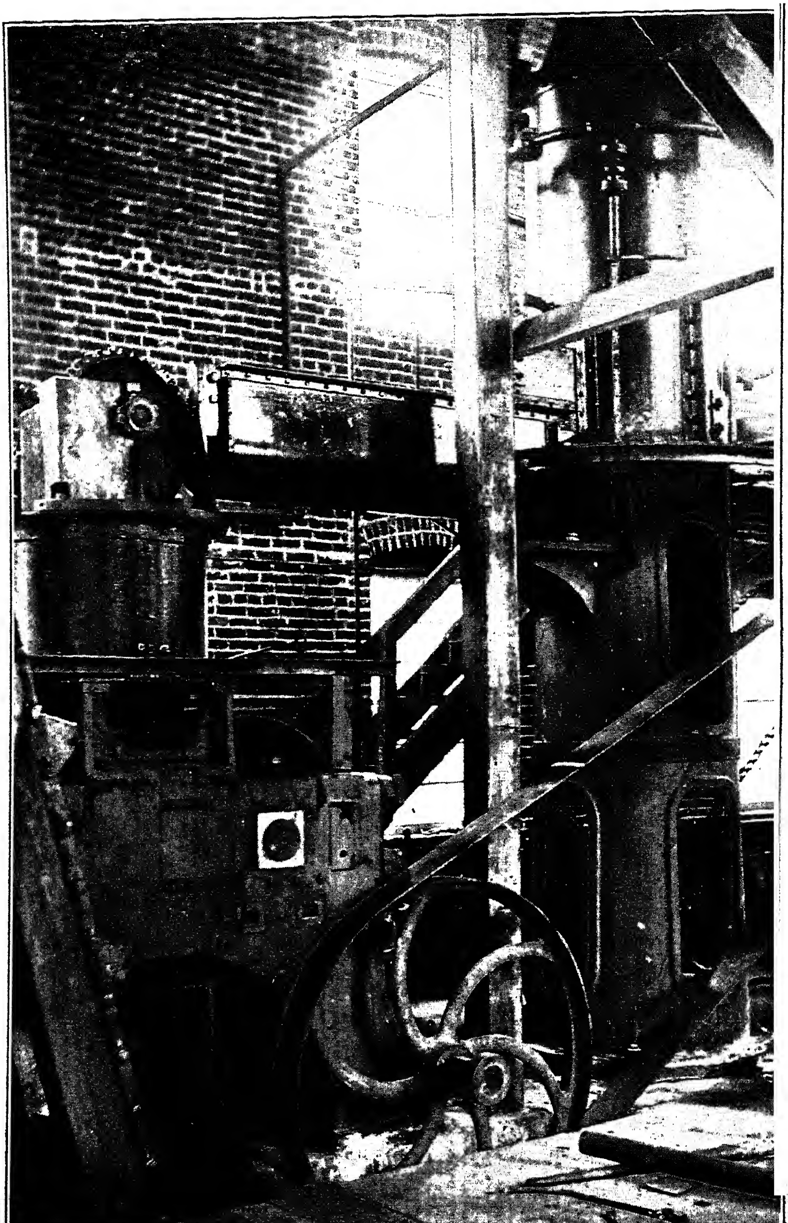

FIG. 16.—VIEW OF BRIQUETTING-PRESS AND MIXING-TOWER.

cents of the estimate of cost of manufacture. Following are the results obtained in the different steps of the process:

Drying.

Approximate capacity of each drier, about 10 gross tons of dry culm per hour.

Moisture evaporated per pound of coal burned, 9 lb.

Coal burned per square foot of grate surface per hour, 8 lb.

Moisture evaporated in percentage of wet material entering the driers, 13.8 per cent.

Power required for drying, elevating, and conveying, 15 kw.

Labor required for drying: 1 fireman at 17.5 cents per hour. 1 laborer unloading culm, 13.5 cents per hour.

Screening and Separating.

Enough No. 2 and No. 3 buckwheat coal is reclaimed from the culm to fire the driers, this material running in sizes: Buckwheat, 60.0; No. 20 mesh, 35.0; and smaller, 5.0 per cent.

The screens give the following results in sizing:

	First Set. Per Cent.	Second Set. Per Cent.	Third Set. Per Cent.	Fourth Set. Per Cent.
No. 10 mesh and larger,	24.7	0.6	0.0	0.0
No. 20 mesh,	70.0	60.5	6.3	0.0
No. 40 mesh,	4.5	34.6	66.2	12.2
No. 60 mesh,	0.6	4.0	24.8	69 4
Smaller,	0.0	0.3	2.6	18.4
	99.8	100.0	99.9	100.0

Depending on the position of the adjustable slate-gate (see Fig. 12), the following results may be obtained in the separators:

	Position No. 1. Per Cent.	Position No. 2. Per Cent.	Position No. 3. Per Cent.	Position No. 4. Per Cent.
Quantity of original culm recovered for briquetting,	90.2	88.5	87.0	80.1
Quantity of original culm refuse,	9.8	11.5	13.0	19.9
Reduction in ash for briquetting-material,	3.7	4.1	4.7	7.4
Quantity of ash of refuse,	65.2	55.2	54.6	54.4

Power required for sizing, separating, conveying, etc., 65 kw.

Labor required, 1 oiler at 17.5 cents per hour.

Mixing of Binder.

No trouble has been experienced in handling and mixing the binder. The pitch has been shipped in bulk loaded in box-cars, and during the summer we may have trouble from softening. The measuring-machine works satisfactorily, and the only labor used is one man at 13.5 cents per hour to feed the pitch to the rolls.

Briquetting.

At first considerable trouble resulted from improper feeding of the mixture to the presses, but this has been eliminated by using 400° superheat for the steam and then cooling the mixture as it leaves the mixing-tower by means of a fan-cooler. An extra elevator had to be built to return the fines which were taken out at the lip-screen of the loading-pocket. Labor required, 2 pressmen at 17.5 cents, and 1 loader at 13.5 cents per hour. Power required, 50 kw.

The whole plant is operated by 10 men, including the foreman, and the total power-consumption is about 135 kw. The hourly production of boulets, or briquettes, from both presses, is from 16 to 17 gross tons.

The Anthracite Board of Conciliation.

BY SAMUEL D. WARRINER, WILKES-BARRE, PA.

(Wilkes-Barre Meeting, June, 1911.)

THE dealings between concentrated capital invested in the conduct of our various industries and the combinations of labor known as "trade union organizations," have produced not only in the United States, but abroad, many novel methods of negotiation between employers and wage-workers.

The size and strength of these organizations have destroyed the personal elements formerly governing the dealings between employer and employee, and have produced various forms of trade-agreements now in more or less successful operation. The difficulty, however, has been to secure consent of both parties to an equitable agreement. For this purpose there have been devised various forms of arbitration, either compulsory under the shadow of governmental legislation, or voluntary by mutual agreement. But these have not been altogether successful from a technical stand-point, for the reason that a compromise has generally been necessary, and in an effort to reach an agreement, terms have been made which have been neither fair nor satisfactory to either of the contending parties.

The Anthracite Board of Conciliation represents one of the few thoroughly successful courts for the settlement of trade-disputes, if not the only one which has yet been evolved. The prestige of this board rests upon its successful treatment of the labor-troubles of the anthracite region for a period of more than eight years, and upon the fact that it has been able

not only to reach decisions based upon the merits of the controversies, but to enforce its decisions upon both the employees and the employers. It has never been obliged to shape its conduct to secure the consent of the contending parties, and its decisions have been strictly and, in the main, cheerfully observed.

The remarkable feature of this board has been that it has been without authority, except the consent of employers and employees to the awards of the Anthracite Coal Strike Commission appointed by President Roosevelt in 1902, backed by the force of public opinion that the procedure established by that commission was a just and effective method of settling labor-troubles.

In 1900 the advent of the United Mine Workers of America (a bituminous labor organization) into the anthracite region, broke a long peace in that region and resulted in a general strike which lasted six weeks. This strike was finally settled by political interference in a manner satisfactory to neither party, and the year of 1901 passed with a feeling of irritation on both sides, marked by sporadic strikes, restlessness, and general dissatisfaction.

In March, 1902, the anthracite workers of the United Mine Workers of America met in convention and passed resolutions demanding recognition of the union, an increase in wages, an 8-hr. day, and the payment of contract-miners of coal by weight, with notice that after April 1 the miners would work only three days per week until the operators would agree to their terms. They further appealed to the National Civic Federation of New York to assist them in securing their demands.

Fruitless meetings were subsequently held by the operators and miners with the Civic Federation, and on May 12, 1902, the mine-workers, by resolution of their executive committee, inaugurated a strike. On June 2, the engineers, pumpmen, and firemen were called out, and, as a result of these orders, nearly the entire body of mine-workers to the number of 150,000 quit work, and remained idle until the strike was called off through the offices of the President of the United States on Oct. 23, 1902.

This strike is perhaps the greatest on record in its duration

and the number of men involved. It was marked by turbulence, rioting, and bloodshed. The calling-out of the firemen and pumpmen made it necessary for the operators to import men to save their mines from being flooded, and in the effort to protect these men the collieries became armed camps, guarded by "Coal and Iron" policemen. It soon developed that the local authorities were powerless, and finally the National Guard of Pennsylvania was called out, and remained in the field until the strike was called off.

The losses of this strike have been calculated to amount to $45,000,000 in business to the operators and fully $25,000,000 in wages to the employees.

In the fall the danger of a coal-famine became so imminent that the President of the United States finally yielded to the importunities of the general public and called into conference the representatives of the operators and the mine-workers. The result of this conference was an appointment by the President on Oct. 16, 1902, of a commission consisting of Brig. Gen. John M. Wilson, E. W. Parker, Judge George Gray, E. E. Clark, T. H. Watkins, Bishop John L. Spalding, and Hon. Carroll D. Wright, Commissioner of Labor, as Recorder.

This commission came into the anthracite region, visited the mines, studied the varying conditions with great care, and later heard testimony from the three contending parties—viz., the union labor, non-union labor, and the operators. A total of 558 witnesses was heard, and finally, on Mar. 18, 1903, the commission reported to the President its findings. This document proved to be successful in settling the controversy, and the Anthracite Board of Conciliation thereby established so far has been successful in carrying out the instructions of the President, "to endeavor to establish the relations between employers and wage-workers in the anthracite field on a just and permanent basis, and as far as possible to do away with any causes for the recurrence of such difficulties as those which you have been called upon to settle."

It is beyond the scope of this article to discuss the various awards of the commission, further than to say that they admirably met the varying labor and physical conditions of the anthracite region, and provided a profit-sharing scheme by which wages automatically advanced or declined with the rise

or fall in the price of coal. It is, however, necessary to explain the fourth and last demand of the miners, because the commission, in making the award under this demand, settled what by general consent was the most important of the demands of the mine-workers, which culminated in the strike; and also because in making an award to this demand the commission provided for the Anthracite Conciliation Board, which is the subject of this article. This demand read as follows:

"The incorporation in an agreement between the United Mine Workers of America and the anthracite coal companies of the wages which shall be paid and the conditions of employment which shall obtain, together with satisfactory methods for the adjustment of grievances which may arise from time to time, to the end that strikes and lockouts may be unnecessary."

This was practically a demand for the recognition of the United Mine Workers of America as a labor organization with which the anthracite coal companies should deal in contracting for their labor. The anthracite companies had consistently declined to deal with this organization or recognize it in any way. Mr. Mitchell, the President of the miners' union, had so far acceded to the operators' position that in his appearance before the commission he acted as the representative of the anthracite coal-mine workers, and not in his official character as President of the United Mine Workers of America. The distinction in theory is perhaps finely drawn, but the practical effect of it has been far reaching in the methods of dealing with labor in the anthracite region.

In its award to this demand the commission delivered what has proved to be an epic among industrial documents, setting forth with convincing logic the relations between employer and worker, defining their mutual privileges and obligations in language which brushed away sentiment and prejudice, and providing a method of dealing which has successfully withstood the test of eight years of trial.

Relative to the rights of the three contending parties, viz.: the non-union employees; the union employees, seeking recognition of their organization and claiming that, if the union employees at any mines are in a majority, such majority shall have the right and privilege of representing and acting for the whole body; the operators, flatly declining to recognize

the mine workers' organization and insisting on the right of dealing direct with their own employees: it is sufficient to say that the award rejected the claim of the organization for recognition, although it favored the practice of collective bargaining along proper lines; upheld the rights of non-union labor, and protected the privileges of employers in dealing direct with their employees; and to the end that strikes and lockouts may be unnecessary, it adjudged and awarded as follows:

"That any difficulty or disagreement arising under this award, either as to its interpretation or application, or in any way growing out of the relations of the employers and employed, which cannot be settled or adjusted by consultation between the superintendent or manager of the mine or mines, and the miner or miners directly interested, or is of a scope too large to be so settled and adjusted, shall be referred to a permanent joint committee, to be called a board of conciliation, to consist of six persons, appointed as hereinafter provided. That is to say, if there shall be a division of the whole region into three districts, in each of which there shall exist an organization representing a majority of the mine-workers of such district, one of said board of conciliation shall be appointed by each of said organizations, and three other persons shall be appointed by the operators, the operators in each of said districts appointing one person.

"The board of conciliation thus constituted shall take up and consider any question referred to it as aforesaid, hearing both parties to the controversy, and such evidence as may be laid before it by either party; and any award made by a majority of such board of conciliation shall be final and binding on all parties If, however, the said board is unable to decide any question submitted, or point related thereto, that question or point shall be referred to an umpire, to be appointed, at the request of said board, by one of the circuit judges of the third judicial circuit of the United States, whose decision shall be final and binding in the premises.

"The membership of said board shall at all times be kept complete, either the operators' or miners' organizations having the right, at any time when a controversy is not pending, to change their representation thereon.

"At all hearings before said board the parties may be represented by such person or persons as they may respectively select.

"No suspension of work shall take place, by lockout or strike, pending the adjudication of any matter so taken up for adjustment."

As a result of this award, the Anthracite Board of Conciliation was first organized in April, 1903, and consisted originally of T. D. Nichols, Wm. Dettrey, and John Fahy, representing the miners in each of the districts into which the region was divided; W. L. Connell, of Scranton, an individual operator; R. C. Luther, of Pottsville, General Superintendent of the Philadelphia & Reading Coal & Iron Co., and myself, representing the operators.

On the death of R. C. Luther some time later, his place was filled by W. J. Richards, the present incumbent.

On the miners' side, Mr. Nichols, who was elected to Congress, was succeeded by Adam Ryscavage, who was later on succeeded by Benjamin McEnaney. Dettrey has been succeeded in turn by John F. McElhenney, John J. Waters, Charles Gildea, and Thomas Kennedy.

The early days of the board were stormy. Neither the operators nor the miners had learned to meet each other on an equal plane, and the method of dealing as laid down by the commission was untried. As a result of adjustments of wages and labor-conditions made by the operators under the award of the commission, a multitude of cases were presented to the board for consideration. Technical questions involving methods of payment under the terms laid down by the commission were raised, which required careful study to determine their merits. Many cases of discrimination by employers against the miners' union were presented. In the early days of the board the settlement of these cases, in spite of an effort at forbearance by both parties, led to much bitterness and mutual misunderstanding. The umpire at first was frequently appealed to for decision, and before him the merits of the controversy were vigorously argued by the contending factions of the board. This state of affairs gradually changed. The miners began to appreciate the desire of the operators' representatives to be fair, and in turn became themselves less aggressive, and more earnest in their desire to be reasonable. The result of this growth of mutual understanding has been a steady decrease in the number of cases presented to the board for adjustment, and an increasing proportion of these cases have been settled directly by the board without appeal to an umpire. In addition, an increasing number of cases have been settled "out of court" by conference between the district representatives of the Board of Conciliation and the parties directly interested.

Upon organization subsequent to the strike, the Board of Conciliation adopted a set of simple rules for the conduct of its business, as follows:

1. If any employee or body of employees have any grievance or complaint growing out of the interpretation of the awards of the Anthracite Coal Strike Commission or out of the application of said awards, or in any way growing out of

the relations of employees and employer, said employee or employees directly interested shall present such grievances to the foreman directly in charge of the mine.

2. If there shall be a disagreement with the foreman, or a failure on the part of the foreman to satisfactorily adjust such grievances, the employee or employees directly interested, or a committee of same, shall request an interview with the superintendent or manager of the mine or mines for the purpose of adjusting said grievances.

3. In case of failure to arrive at a satisfactory adjustment of grievances, the employees shall present in writing such grievances to the member of the Board of Conciliation representing the district in which the mine or mines are located, stating fully the grievance which they desire to have adjusted, and offering satisfactory proof that efforts have been made to arrive at an adjustment with the superintendent or manager of the mine or mines.

4. In case of a failure on the part of the superintendent or manager of the mine or mines to grant an interview with the employee or employees within ten days, said employees may present in writing to the member of the Board of Conciliation representing their district proof that they have made reasonable efforts to secure such interview. In such case the Board of Conciliation, or the members of the board representing the said district, will endeavor to secure for them an interview with the superintendent or manager of the mine or mines in question.

5. The board will act upon the grievances presented to them in accordance with the above rules by notifying the company or operator with whom such difficulty or disagreement may arise, and requesting from him a statement setting forth his reasons for not adjusting such difficulty. After receiving such statement the board will, if necessary, at its discretion, request the presence of both parties to the disagreement for a full and complete hearing of the case.

6. In case of any complaints or grievances which may arise on the part of employers, the said employers having such grievance may present the same to the member of the Board of Conciliation representing the district in which the mine or mines are located, and the board will receive such complaints and call for a statement from the employees of said mine or mines relative to the reasons for such complaint or disagreement, and if in its judgment such action is necessary, will request both parties to the issue to be present for a hearing of the case

7. Inasmuch as the Anthracite Coal Commission in their award have provided that no suspension of the work shall take place pending the adjudication of any matter brought before the board for adjustment, and to the end that no strikes or lockouts shall be necessary, the Board of Conciliation will not take up and consider any question referred to it unless the employees shall remain at work, with the understanding that if the said board shall decide that the grievances are justifiable, the adjustment shall be retroactive

8. Whenever there is an accumulation of cases presented to it for decision, the board shall continue in session until such cases are disposed of so far as practicable.

9. It shall be the duty of the respective representatives of the operators and miners in each district to endeavor to settle grievances before they are formally presented to the board, but failing to do this, the grievance shall be filed with the secretary of the board, who shall thereupon send out a copy of said grievance to each member of the board, and also a copy to the defendant, with a request for an answer thereto as soon as possible.

10. The board will then set a date for the hearing of said grievance, and notification will be sent to both parties of such date and place of meeting, at which testi-

mony will be taken. It will be incumbent on both parties to appear, unless excused by the board for sufficient cause.

11. In case a complainant fails to appear, or fails to corroborate his statement of the grievance by sufficient testimony, the case will be promptly withdrawn from before the board, so that the files of the board may be cleared.

12. In case a defendant fails to appear or fails to present a defense, the board will assume as correct the statement of the grievance and testimony thereto as made by the complainant

13. If a grievance is withdrawn or not sustained for lack of sufficient evidence, a similar grievance may be filed subsequently with the board without prejudice, it being understood that the retroactive rule of the board in the adjustment of such grievance shall date from the filing of the grievance which is subsequently sustained.

One of the most important features of these rules was a provision for the retroactive settlement of grievances. The effect of this was to allow plenty of time for the consideration of grievances, and in case a grievance was found to be justified and it involved a loss of pay to the employee, the settlement started with the date at which the grievance was formally presented to the Board of Conciliation.

Many of the decisions of the Board of Conciliation have provided for this retroactive feature, and the effect of it has been to impress upon the employees the uselessness of strikes, and the certainty, in case the grievance for which they were complaining was justified, that the settlement would take effect without loss of time. To the Conciliation Board it was equally important in giving it plenty of time to consider the case thoroughly, and in many instances a much fairer solution was reached after the bitterness of the grievance was somewhat lessened.

The hearings of the board are conducted in a somewhat informal manner as compared with court procedure. The witnesses (many of whom are foreigners) are allowed as great latitude as possible in the statement of their troubles, and although the general rules of the board have been lived up to, yet it is the practice of the board not to confine itself strictly to technicalities, viewing the broad merits of the case, in an effort to arrive at a decision fair and just to both contending parties. This course has been beneficial to the general labor situation. In many instances men have come before the board, filled with grievances which have aroused their passions, and unable to listen to the position of the other party, and after the heat of passion has cooled off, are often quite willing to admit

that their original position was hastily taken, and that the position of their opponents was in a sense justified. One case in particular may be cited as illustrative: that of a man working for one of the larger anthracite companies, who presented a grievance to the board, and whose testimony was later heard. After an hour or more of listening to his testimony, it developed that the man was suffering from a general feeling of irritation, but had no specific demand to make, and he was flatly asked just what he wanted of his employer. His answer was: "I don't want anything; I just came before you people to show the venomy with which I was treated; I have a better job now than the one at which I was working, and would not go back to work under any condition." He later left the board in good humor. This case is cited as illustrative of the puerile causes which sometimes have led to general strikes. Until this man's testimony was heard by the board, he had been a trouble-breeder at the colliery, and a general feeling of irritation was present among the employees.

The authority of the board has been severely tried by important cases involving questions of wages and discipline, at not only individual collieries, but groups of collieries operated by the larger companies, and in one or two instances, in spite of the rules of the Conciliation Board and of the Anthracite Coal Strike Commission, serious strikes of short duration have occurred. Perhaps the most serious one in the history of the board was that of the employees of the Pennsylvania Coal Co., which involved 10 collieries of that company. At these collieries the payment for the mining of coal by contract-miners was made by weight, under an old basis in vogue in the region for many years, known as the "miner's ton," amounting to about 2,700 lb., which represented the number of pounds of raw product which was necessary to make one ton of coal of the prepared sizes of chestnut and larger shipped to market, the miner being paid a fixed sum for this amount. Owing to changes in trade-conditions, the miners believed that they were treated unfairly in the calculation of the "miner's ton," and on account of there happening to be a large percentage of ignorant Italian labor at these collieries, local agitators got to work among them, and, in spite of the efforts of the company, a bitter strike resulted. For a time it looked as if this strike

might spread throughout the region, but the matter was finally settled through the influence of the Conciliation Board. After an agreement to submit the differences to the Conciliation Board had been secured, the matter was amicably adjusted by the board after a full hearing of testimony from both sides.

Up to date, a total number of 192 grievances have been presented to the board. Of these, 180 grievances are of employee against employer, 1 grievance of employee against labor organization, and 11 grievances of employer against employee. Statistics show that on the employee *vs.* employer grievances, the following action was taken: Sustained, 15; not sustained, 33; settled by agreement of parties, 31; partly sustained, 31; no jurisdiction, 9; withdrawn for lack of sustaining testimony, 52; pending, 9. Of the employer *vs.* employee grievances, 2 were sustained, 1 settled by agreement of parties, 6 withdrawn, 1 no jurisdiction, and 1 is pending. Altogether, 25 cases have been referred to an umpire, as provided by the commission.

In the early years of the board a great many grievances were cases of discrimination for or against labor unions. Under the award of the Anthracite Coal Strike Commission, the equality of labor, whether union or non-union, was maintained, and it was adjudged that there should be no discrimination for or against labor unions. Troubles between foremen or superintendents, who were sometimes too antagonistic against labor unionists, and employees who were too radical or insistent in enforcing union methods in the dealings between employers and employees, produced many of these cases, and finally led to a decision of the Board of Conciliation which sustained the absolute right of the employer to hire or discharge his labor, provided there was no discrimination because of membership in a labor organization. From the time of this decision a great many of the cases of alleged unjust discharge, in which the discharged employee believed that he was discharged on account of membership in a labor organization, ceased, and for several years no cases of this kind have been presented to the board.

In the later years of the board the majority of cases have related to the subject of wages. The award of the Anthracite Coal Strike Commission provided for a schedule of wages

which would be unchanged, provided the conditions of labor were to remain unchanged. This agreement was reaffirmed in conference between the operators and miners three years and six years later. The anthracite region, however, is subject to varying conditions of labor, especially with contract-miners, and in this respect is very different from the bituminous region. The result is that hardly two collieries have the same method of adjusting wages. The seams of coal pitch at all angles from flat to vertical and vary greatly in thickness and quality. The conditions are constantly changing, not only on account of the greater extent of the workings, but also on account of new seams being opened up and mined. On account of these new conditions of employment, it has become one of the most important functions of the Board of Conciliation to adjust the terms of payment, so that the wage-earning capacity of the employee may remain unchanged. In other words, so that for the same unit of labor he may receive the same unit of price as formerly. This, of course, is a very easy matter if conditions are unchanged, but with the opening up of new mines and new work it has become necessary to establish new prices for mining coal and for yardage, as well as to provide proper adjustments for impurities in the seams, and in many instances the Board of Conciliation has been called upon to fix new prices for work of this description, and in fact adjust the entire wage scale at new collieries, so that these prices and wages may compare equitably with corresponding work in that region. The proper consideration of these grievances has required the Board of Conciliation not only to hear complex testimony, but also to visit the mines and study the conditions upon the ground.

After eight years of experience, the work of the Conciliation Board may be summed up as follows:

From the operator's stand-point it has been a good business investment in securing for him freedom from losses due to strikes, and protection from extravagant demands of his employees.

It has also provided the operator with a channel by which he can more readily reach the heterogeneous class of employees of many nationalities and speaking many languages which now find employment in the anthracite region. The

miners, through their organization, can control this class to better advantage and prevent many troubles which are primarily due to feuds and disturbances among the men themselves.

From the stand-point of the non-unionist, the results are beneficial in that it has secured for him freedom from the tyranny of labor organizations, full protection for himself and family, and absolutely fair and equal treatment as compared with unionists.

From the stand-point of the labor unionist the results have not been, perhaps, as satisfactory as he would wish. Organizations of the character of the United Mine Workers of America, having in their ranks members speaking different languages and made up so largely of foreigners, thrive largely on agitation, and it is only at times of strikes that the leaders are able thoroughly to coalesce the men together and secure from them payment of dues. The peaceful conditions which have prevailed in the anthracite regions for so long have not been conducive to a strong membership in the union, the men feeling that they are protected in their employment regardless of their membership in a union, and, with natural economy, are loath to continue the expenditures necessary to renew their membership. While this condition is of advantage to the operators in giving them the benefits of an organization with which to do business, and at the same time keeping this organization within the bounds of reason and preventing it from becoming radical on account of its very strength, yet from the standpoint of the miners' representatives there has been more or less serious complaint regarding the injustice of their being asked to represent the whole body while they are paid only by their own constituents. The result of this is that at times the miners' representatives on the Board of Conciliation have had great difficulty in securing the co-operation of their constituents in carrying out the decisions of the board and have had occasionally to go to great lengths to prevent trouble. Yet all in all, taking into account the ignorant condition of many of the employees, and the great danger to the trade that would ensue if this heterogeneous mass were to secure a strength that would come with a larger membership and treasury, with the inevitably resulting exorbitant demands and strikes

that would occur, it cannot be denied that from the stand-point of the public the work of the Conciliation Board has been beneficial in securing a period of peace and prosperity uninterrupted by danger of coal-famine due to strikes, and in general all of the benefits which come from an even regulation and conduct of the business. The local merchant and landlord, and even the local press, influenced as it is by the necessity of catering to the labor element, have unanimously indorsed the board as a potent influence in preserving the regularity of payroll disbursements, in keeping good order, and in furthering industrious habits and more peaceful conditions. As a result of such indorsements the public demand has largely strengthened the authority of the board in its work, and prevented so far any successful move for its abolition by the radical elements.

Lead-Smelting in the Ore-Hearth.

BY J. J. BROWN, JR.,* WILBURTON, OKLA.

(Wilkes-Barre Meeting, June, 1911.)

THE ore-hearth was the earliest type of furnace used in smelting Mississippi Valley lead-ores, which are very pure, and low in silver-content. The first smelters made no attempts to recover lead from the smoke; and since about 15 per cent. of the lead in the charge escaped in this manner, early practice with the hearth was decidedly wasteful. At most of the large smelting-plants blast-furnaces with auxiliary roasters were substituted for the hearths long ago. But, upon the introduction of the Lewis and Bartlett bag-process for collecting fume, the ore-hearth was used by companies engaged in the manufacture of pigments, simply on account of the large percentage of fume made by it. In fact, the extraction of lead was kept down by the use of hot blast and other devices.

During recent years, however, several smelting companies have realized that the preliminary roasting of galena for the blast-furnace is not economical, and have, therefore, replaced the roasting-furnace with a modification of the old Scotch hearth, in which a large percentage of the lead in a charge is

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recovered directly as pig-lead, while the remainder passes partly into fume and partly into slag low enough in sulphur to be charged into the blast-furnace without further treatment. The results obtained have proved so satisfactory, both as to recovery and operating-expense, that one eminent metallurgist has predicted the universal use of the ore-hearth on non-argentiferous ores.

The ore-hearth, thus employed as an adjunct to a blast-furnace plant, is rather a desulphurizer than a smelting-furnace proper,—the chief object being to make a blue slag suitable for the cupola-furnace. Hence, no special care is taken to obtain from it a large lead-extraction.

But the Granby Mining & Smelting Co., at Granby, Mo., having no blast-furnace to handle the slag produced (which it sells to other smelters), aims to obtain a large lead-extraction, and a slag carrying the minimum percentage of lead. To promote this end, the smelter-men are paid in proportion to the number of pounds of metallic lead which they produce from a given charge.

The total charge per hearth per day weighs 14,000 lb., and consists of galena-concentrates and blue and white fume, in variable quantities. The blue fume is collected in a steel chamber adjoining the line of hearths, while the white fume is filtered in cotton bags by the Lewis and Bartlett process.

When I went to Granby, about five years ago, the plant was equipped with the old water-backed Scotch hearths, 2 ft. wide, burning charcoal as fuel, and smelting 14,000 lb. of charge in about 14 hr. This period was divided into two shifts, four men to the shift, who worked in pairs, relieving one another every 15 or 20 min. Two yard-hands supplied five furnaces with their charges, fuel and lime, and removed the pig-lead as molded.

The smelters were required to convert into pig-lead 70 per cent. of the galena charged, 50 per cent. of the white fume, and 40 per cent. of the blue fume. For each pound of lead made in excess of these percentages, they were allowed one cent (divided among eight men), in addition to their daily wage of $2. If their extraction fell under the above-named percentages, they were penalized in like proportion.

Under this system a very high extraction in metallic lead—

averaging about 85 per cent. of the lead-content of the charges —was obtained. Only about 2 per cent. went into the blue slag, while the remainder was recovered as fume, and resmelted.

Having formed the opinion that it would be advantageous to sacrifice a little in high initial extraction, and thereby to reduce materially the cost of smelting, I obtained permission to experiment on a 5-ft. air-backed Jumbo hearth, which had been abandoned as less efficient than the Scotch hearths. The results were so satisfactory that all the Scotch hearths have since been replaced by Jumbos, which have not only proved more profitable to the company, but also permit the workmen to earn a third more wages with a third less work in hours.

The percentages required of the smelters were changed to 65 per cent. for galena, 60 per cent. for white fume, and 50 per cent. for blue fume, based on the weight of charge,—the excess or deficiency on these percentages, as the case might be, being divided among four, instead of eight men, as formerly.

The lead-extraction for the six months ending Dec. 31, 1906, based on the lead-content of the charges, proved to be only 2 per cent. less on the Jumbo hearth than on the Scotch hearth.

In view of these facts, and also of the use of the cheaper fuel, —bituminous coal, instead of charcoal—the following statement, based on results obtained during the aforesaid six months, will be readily understood.

Results on Three Jumbo Hearths.

Lead-content of 5,425,000 lb. of galena-concentrates, white and blue fume, and dry bone smelted, ... 4,304,369 lb.

Material recovered:

Pig-lead, ...	3,568,112 lb. at	$5.80 per cwt.,	$206,950.50
Slag (38.8 per cent. Pb),	296,500 lb. at	1.60 per cwt.,	4,744.00
White fume, ...	677,300 lb. at	4.00 per cwt.,	27,092.00
Blue fume, ...	190,404 lb. at	3.50 per cwt.,	6,664.14
Total, ...			$245,450.64

Value of recoveries per 1,000 lb. of materials smelted, $45.2443

Smelting-expense:

Labor (smelters, yard-hands, etc.), ...	$5,921.80
Fuel (1,552 bu. charcoal at 8 cents, and 4,313 bu. stone coal at 10 cents), ...	555.46
Lime (614 bu. at 20 cents), ...	122.80
Total, ...	$6,600.06

Cost per 1,000 lb. of materials smelted, ... $1.2166

Results on Two Scotch Hearths.

Lead-content of 3,591,000 lb. of galena-concentrates, white and blue fume, and dry bone smelted, ... 2,848,942 lb.

Materials recovered:

Pig-lead, ...	2,462,212 lb. at	$5.80 per cwt.,	$142,808.29
Slag (39.4 per cent. Pb),	189,650 lb. at	1.60 per cwt.,	3,034.40
White fume, ...	340,170 lb. at	4.00 per cwt.,	13,606.80
Blue fume, ...	95,636 lb. at	3.50 per cwt.,	3,347.26
Total, ...			$162,796 75

Value of recoveries per 1,000 lb. of materials smelted, $45.3346

Smelting-expense:

Labor (smelters, yard-hands, etc.), ...	$5,044.45
Fuel (10,122 bu. charcoal at 8 cents), ...	809.76
Lime (352 bu. at 20 cents), ...	70.40
Total, ...	$5,924.61

Cost per 1,000 lb. of materials smelted, ... $1.6496

Comparing the expenses and recoveries of the two types of ore-hearths, as detailed in the above statement, we find that there was an advantage in favor of the Jumbo hearth of $0.3427 per 1,000 lb., or $0.6854 per ton, of material smelted, or more than $1,200 per hearth per year. This does not include the decreased engine-room and bag-house expense, of which I will speak later.

In order that the above comparison should be correct in full, precautions were taken that the charges on all hearths, both Scotch and Jumbo, were identical each day during this period.

It will be observed that the amount of lead converted into blue slag was practically the same for each type of hearth, the difference being less than 1,000 lb. per hearth for the entire six months.

Another important economy of the Jumbo hearth is, that it permits a charge to be smelted in much less time, and thus saves labor and expense as regards water, blast, and the bag-house. The difference at Granby was found to be from 4 to 5 hr. a day. This is due to the larger fire-area of the Jumbo hearth. The larger the hearth, the more material can be smelted at one time, providing the smelter-men are able to perform the increased amount of work entailed.

It was found that a higher extraction was obtained on the 4-ft. than on the 5-ft. hearth, for the reason that the latter had a little too much fire to be worked to advantage, although it required less time to run out a charge.

The advantage of the water-back over the air-back is two-fold. Accretions are much less likely to form on the water-back; and it has a tendency to hold down the percentage of lead passing off as fume; or, as we say, it "burns up less lead." The air-backed furnace has the same kind of effect in this respect that a hot blast would have, although to a less degree.

The results in smelting "dry bone," or cerussite, are much more satisfactory on the larger furnaces; in fact, the company had ceased smelting it until they were installed. This may be accounted for by the fact that the carbonate ore requires for reduction more heat than galena; and a hotter fire can be maintained on this furnace than on the old type.

The shape of the basin seems to make considerable difference in obtaining a high extraction of lead. The usual pattern has a front sloping about 65°; but I believe that if the front is vertical, the browse does not pile up so much at the edge of the working-hearth, and the work of the smelters is lighter. At least, this has been the experience at Granby.

The following statement shows the actual smelting-results for the six months period ending Dec. 31, 1906:

Materials Smelted.

Material.	Pounds.	Lead. Per Cent	Water Per Cent.	Lead-Content Pounds
No. 1 mineral, . .	7,717,650	82.4	1.909	6,237,942
White fume, . . .	910,200	72.8	Dry	662,625
Blue fume, . . .	259,600	67.3	Dry	174,711
Dry bone, ...	128,550	63.1	3.8	78,035
Total, ...	9,016,000			7,153,311

Materials Recovered.

Material.	Pounds.	Lead. Per Cent.	Water Per Cent.	Lead-Content. Pounds.
Pig-lead, ...	6,030,324	......	......	6,030,324
White fume, . . .	1,017,470	72.8	Dry	740,718
Blue fume, . . .	286,040	67.3	Dry	192,505
Slag, Scotch hearth,	189,650	39.4	Dry	74,722
Slag, Jumbo, . . .	296,500	38.8	Dry	115,042
Total, ...	7,819,984			7,153,311

It will be observed that all of the lead was recovered, except 2.653 per cent., which entered the blue slag. Or, to be more exact, the total lead-content of the charges smelted was distributed as follows:

	Per Cent
Recovered directly as pig-lead, ...	84.301
Recovered directly as white fume, ...	10.355
Recovered directly as blue fume, ...	2 691
Recovered directly as blue slag, ...	2.653
	100.000

As already observed, the Granby Co. sells its blue slag to the highest bidder, having no slag-eye, or shaft-furnace, in which to smelt it; but we may safely assume that not more than 0.5 per cent. of the lead in this slag is finally lost in its re-treatment. On this assumption, we find that only 0.132 per cent. of the lead in the original charges was lost; in other words, the total extraction was 99.868 per cent.

It should be emphasized that, in order to obtain this high extraction, the cost of smelting was not increased out of proportion to the amount of lead saved. This item, exclusive of the cost of smelting the slag, was $3.95 per ton of charge treated. For each ton of charge there was 0.02105 ton of blue slag produced, and this slag can be smelted for $4.50 per ton, or $0.095 for the quantity named. Hence the total cost of treatment would be only $4.045 per ton. This figure could be much reduced if smelting were conducted on a larger scale.

Naturally, the heat is intense in front of the hearth—more so on the large Jumbos than on the smaller Scotch hearths; and therefore a blast of air is kept continually playing on the backs of the smelter-men. There is some difference of opinion as to whether this air-blast should be delivered near the floor, and allowed to ascend, or whether it should be delivered above the heads of the workmen and directed downward. I hold the former opinion, believing that the cooler air from the blast-pipe, if delivered near the floor, tends to lift the heated air away from the workmen, while a downward blast tends to hold the hot air where it is least desired.

Some ores are much more difficult to smelt with a good extraction than others. They require hotter fire, more labor in stirring the fire, etc. Therefore, it is necessary for the work-

men to be able to regulate the blast to suit themselves. The blast-pressure is seldom greater than 12 oz. in any case.

Much better results can be obtained from coarse than from fine ore, since it is not so readily blown or drawn into the trail. If the ore be too coarse, however, it remains too long on the fire before fusion. Pea-sized is more satisfactory than any other.

When a proportion of "dry bone" (lead carbonate) is to be treated, it is advisable to reserve this material until all of the galena and fume has been smelted, and then, when the furnace is hottest, run it through as quickly as possible, because, unless a great deal of fuel is used, the fire soon cools. This is due to the absence of sulphur in the carbonate, and to the negative heat-reaction of its reduction.

The greatest objection to the employment of ore-hearths in lead-smelting is, that the furnaces are too small to permit the workmen to treat a large quantity of ore. The only way to overcome this drawback is to employ mechanical means to work the fire.

Since writing the foregoing paper, I have secured letters patent from the United States and the Dominion of Canada for a furnace in which the fire is worked mechanically, by means of rakes or rabbles. This furnace may be from 10 to 20 ft. wide, instead of from 20 in. to 4 ft., like those now in use; and it can be operated by one-half the number of men required by the small hearth. Moreover, by virtue of the increased hearth-area and daily capacity, the amount of fuel per ton of ore will be proportionally decreased. These several factors should effect an economy of at least $1 a ton over the results as outlined above.

At some future time I may be able to report the results of commercial practice with this invention. Meanwhile, for the information of those members of the Institute who may desire to study its form and principle, I refer them to U. S. Patent No. 888,582, dated May 26, 1908, for "Roasting and Smelting Furnaces," and to the Canadian patent No. 118,913, dated June 15, 1909.

The Caddo Oil- and Gas-Field, Louisiana.

BY WALTER E. HOPPER, ITHACA, N. Y.

(Wilkes-Barre Meeting, June, 1911.)

I. Location and Extent.

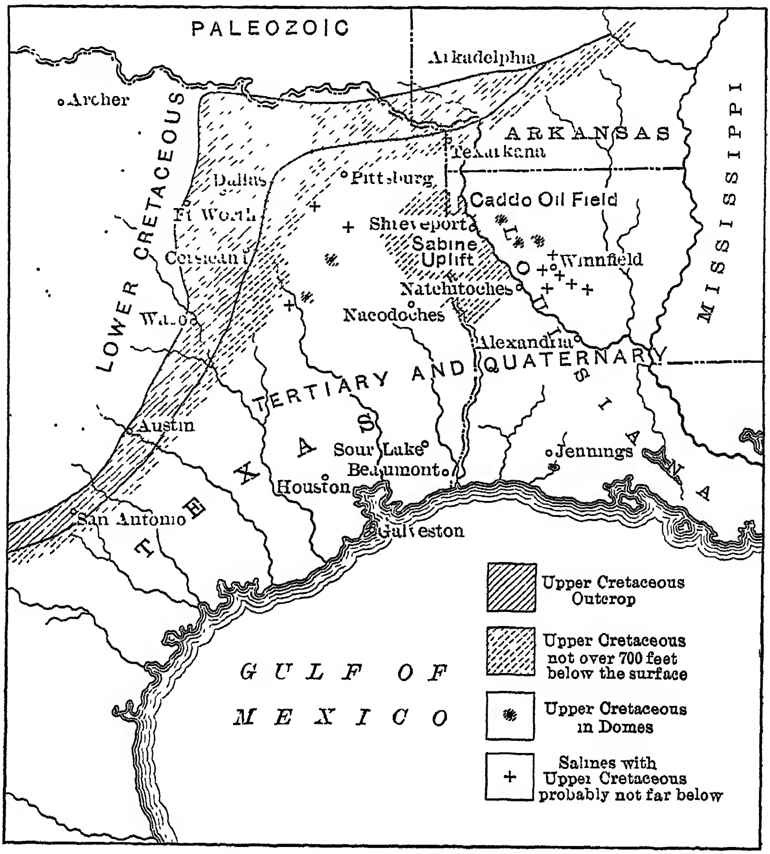
The Caddo oil-field, shown in Fig. 1, is located in Caddo parish, northwestern Louisiana. The known producing territory of oil is covered by townships 19 N, 20 N, 21 N, 22 N, and ranges 15 and 16 W., shown in Fig. 2. The center of the field may be taken as Oil City, 24 miles north of Shreveport, on the Kansas City Southern railway. Shreveport is the second largest city in the State, and is the connecting point of five railroad-lines. Drilling at the present time, however, is going on all over northern Louisiana, especially in Caddo and the neighboring parishes. During the winter of 1908–09 I spent four months in the Caddo field, under the direction of the Louisiana Geological Survey and the U. S. Geological Survey.

II. History and Development.

Natural gas escaping at the surface is found at numerous places in northwestern Louisiana. At Shreveport, the plant of the Shreveport Ice & Refrigerating Co. has been lighted by natural gas for 20 years. The well was drilled for water, but was abandoned on account of the gas. A test well, put down in 1905 near the western limits of Shreveport, was driven to 1,650 ft., and encountered indications of gas and oil at various depths, but did not succeed in finding enough to be profitable. Attention was first attracted to the Caddo field in 1895 by indications in water-wells from 40 to 60 ft. deep, in which the pressure of natural gas was noticeable. This indication of gas in a shallow well led to the drilling of the first well in the Caddo field, the old Savage Brothers & Morricell, or the Caddo Lake Oil and Pipe Line No. 1. The rig for this well was erected in May, 1904, and drilling began in June, 1904. The well was bailed Mar. 23, 1905, with a small amount of oil. It was deepened July, 1905, and converted into a "gasser" Jan. 3, 1906. It was abandoned January, 1907.

In consequence of the finding of oil in the Savage well, drilling-operations were pushed with energy; and in April, 1905,

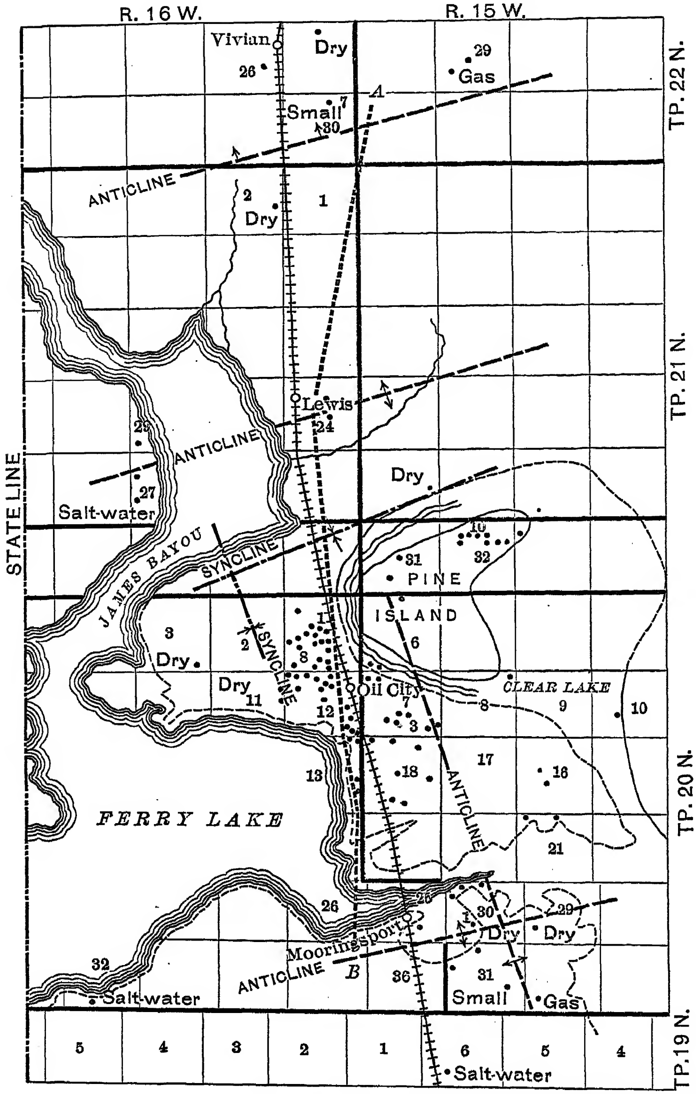
four wells were drilling. Great trouble was experienced with gas blow-outs and several of the rigs were shut down. Difficulty in obtaining wood for fuel during bad weather led to the abandoning, so far as oil was concerned, of the Producers' No. 1, which was then used to supply fuel-gas to the various drilling-rigs in the field.



From *Bulletin No.* 8, *Louisiana Geological Survey* (1909).

FIG. 1.—MAP SHOWING LOCATION OF CADDO FIELD IN REFERENCE TO THE SABINE UPLIFT.

On the afternoon of May 7, 1905, the Producers' No. 2 blew out. The roar of the gas was heard 14 miles away. On the afternoon of May 8 the ground near the hole caved in, taking with it a steam engine, a rotary drilling-rig, a 70-ft. derrick, a hoisting-drum and steel cable, two Gardner pumps (10 by 6 by 10 in.), a tool-house (10 by 12 ft.) with full assortment of tools,



From *Bulletin No.* 8, *Louisiana Geological Survey* (1909).

Vivian region : Little oil, heavy ; much gas. Nacatoch horizon.
Lewis region : Little oil, heavy ; much gas. Nacatoch horizon.
Pine Island : Gas from the Nacatoch sands ; heavy oil and gas from the Woodbine.
Oil City, Sections 1 and 12 : Gas from the Nacatoch, Austin, and Woodbine ; light oil from the Austin, locally ; heavy oil from the Woodbine.
Oil City, south of, Sections 13 and 18 : Gas in the Nacatoch and Woodbine ; light and heavy oil in the Woodbine.
Mooringsport : Gas in the Nacatoch and Woodbine ; light oil in the Woodbine, Sections 26, 25, and west 1–2 of Section 30.
James Bayou, west : Light oil in the Woodbine. Gusher territory.

FIG. 2.—OUTLINE-MAP OF CADDO FIELD, SHOWING STRUCTURE AND AREAS OF GREATEST DEVELOPMENT.

150 ft. of 10-in. casing, 400 ft. of 8-in. casing, 1,600 ft. of 6-in. casing, and 1,600 ft. of 4-in. drill-pipe. A new location was immediately made 300 ft. NE. of the blow-out.

Drilling progressed slowly on account of the blow-outs. On June 18, Producers' No. 2 was lighted, the flames rising 100 ft. In July, 1905, one gas- and one oil-well were producing, eight were drilling, and seven new rigs were up. On Oct. 12, 1905, the contract was made to lay a 6-in. gas-line 23 miles long, extending from the Caddo field to Shreveport, La. Four good "gassers" were then producing. In November, Producers' No. 3 blew out, and in December it caught fire, producing similar results to Producers' No. 2, which was on fire from June to November, when it choked up and died out. On Dec. 31, 1905, four "gassers" had been completed in the field; two wells had been completed for oil, but were not producing; three had been drilled and abandoned; three were drilling; and six locations had been made for new holes.

In February, 1906, the Citizens' Oil and Pipe Line No. 1, in the south end of the Gilbert farm, blew out, throwing mud and water from the hole, 10 ft. wide on the outside of the casing, 100 ft. into the air. Pure gas in strong force came out of the 6-in. pipe. The well finally engulfed derrick, etc., and converted itself into the well-known oil-and-gas geyser. The well was set on fire and is still burning.

In the early part of 1906 the 6-in. gas-line from Caddo to Shreveport was completed. On May 24, natural gas from the Caddo field was made available for domestic consumption. The daily consumption in Shreveport was about 5,000,000 cubic feet.

During the latter part of the year several wells were drilled on Pine Island. In December, Producers' No. 1 on Pine Island was making 250 barrels. A site for a pump-station and tank-farm was acquired at Caddo City, and the laying of a 6-in. pipe-line to Pine Island, about 2.5 miles, was begun. Loading-racks were erected on the Kansas City Southern railway. The first car-load of Pine Island crude oil was shipped from Caddo City to the refinery at Port Arthur on Dec. 13. Seven cars, or 1,510 barrels, had been shipped up to the close of December. The Caddo field produced 4,650 barrels in 1906. Several good "gassers" were also completed during the year.

Considerable drilling was done on Pine Island and north of Oil City during the early part of 1907. On May 23, a deep oil-

well near the old Savage well was brought in, producing 100 barrels flowing. In June, 1907, there were ten big and one small "gassers" completed in the field. The ten wells varied in capacity from 8,000,000 cu. ft. to 25,000,000 cu. ft.; the aggregate daily capacity of the ten was put at 143,000,000 cu. ft. On Sept. 14, 1907, a second gas-line from the Caddo field to Shreveport was completed. During the year 1907 eight producing wells, 11 "gassers," and four dry holes were completed, and nine holes were abandoned.

The year 1908 marked great activity in the Caddo field. Wells were put down north, south, and east of Oil City. In the spring an 8-in. gas-line was started from the Caddo field to Texarkana, Tex., a distance of 55 miles. In April, ten wells were flowing and producing oil. Several good "gassers" had been brought in at Dixie, La., 6 miles SE. of Oil City.

Mooringsport, 4 miles south of Oil City, was opened up with "gassers," and the Hostetter No. 1 in April. On May 11, 1908, the C. G. Dawes Trustee No. 1 blew out; the gas caught fire, and the well burned until Feb. 12, 1909.

A good oil-well was "drilled in" on Pine Island in May, 1908, and several were drilling. The field was considerably extended by the bringing in of a 50,000,000-cu. ft. "gasser" at Lewis, 4 miles north of Oil City, in July. Development was also started at Vivian, La., 10 miles north of Oil City. In September, the best well yet completed in the field was brought in. This is the Producers' Lane No. 1, south of Oil City. It yielded from 2,500 to 3,000 barrels a day at the start, and produced 1,000 barrels daily for several months. In December, the Gulf Refining Hostetter No. 2 was brought in, producing 300 barrels, and calling everybody's attention to Mooringsport. On Dec. 15 there were 25 wells drilling. During the year 1908 Caddo completed 43 producing oil-wells, eight "gassers," and seven dry holes; and two holes were abandoned.

In 1909 the output of the Caddo field was 985,226 barrels, as compared with 499,937 barrels the year before. In January the production averaged 2,177 barrels a day, and it gained at intervals until in November, when a deep well in new territory, 5 miles NW. of the nearest producers, came in, flowing more than 3,000 barrels a day of 40-gravity oil, the output went up to 3,711 barrels a day. During 1909 a 6-in. pipe-line was laid from the Caddo field to a refinery on Grigsby's Island, south of Shreveport.

During the year 1910 almost international attention was attracted to the Caddo field because of a number of large gushers of high-grade oil. During the year nearly twice as many wells were completed, with an initial flow almost 15 times as great as in 1909. The total production during 1910 reached 5,680,000 barrels. For several months it was maintained at a rate of 20,000 barrels a day, but at the close of the year it declined to 14,000 barrels a day.

Three pipe-lines were laid from the field to the Gulf and to the Mississippi river. The Standard Oil Co. of Louisiana established a trunk-line connection with the field to the refining-plant at Baton Rouge. Late in the year the Gulf Pipe Line Co. installed a 6-in. line from the west side of the field connecting with its 8-in. Oklahoma trunk-line to Port Arthur. The Texas Co. completed an 8-in. line direct from Shreveport to its large station at Garrison, Tex. This company also established a storage-tank farm near Shreveport, where more than 1,000,000 barrels was accumulated prior to the completion of the pipe-line. Late in 1910 the Standard Oil Co. took over the holdings of the J. C. Trees Oil Co., at a price believed to be about $4,500,000. Prospecting is going on in every direction to discover more gusher territory of the light paraffin oil.

Final arrangements have been made for the natural-gas line from the Caddo field to St. Louis, and the work preliminary to construction is now under way. A 20-in. line is contemplated, and it is the plan to lay it as nearly on an air-line as possible, in order to save distance. There will be five compressor-stations, from 60 to 75 miles apart. The length of the line as indicated by the preliminary surveys will be 450 miles. The line will have a capacity of 80,000,000 cu. ft. per day.

The 16-in. line to be built by the Arkansas Natural Gas Co., surveys for which are now being made, to extend from the Caddo field to the larger cities of Arkansas, will have a maximum capacity of 30,000,000 cu. ft. daily. It is the general impression at present that the proposed line to New Orleans, a distance of 400 miles, will never take form. However, the gas-line from the Caddo field to Houston, Tex., is almost a certainty. With all the various lines in operation that are proposed, and including the present consumption, it is estimated that the average quantity of Caddo gas consumed will be more than 125,000,000 cu. ft. per day.

The production of the Caddo field is given in Table I.

TABLE I.—*Production of the Caddo Field for the Years 1906, 1907, 1908, and 1909.* [a]

1906.

December 14—250 barrels. December 31—150 barrels.

1907.

Month.	Producing Wells Completed.	New Production	Gas.	Dry.	Abandoned
January, ...	1	100	0	0	0
February, . . .	0	0	0	3	0
March, ...	0	0	2	0	1
April, ...	0	0	3	0	0
May, ...	1	150	0	1	2
June, ...	0	0	2	0	4
July, ...	3	305	0	0	0
August, ...	0	0	1	0	1
September, . . .	0	0	1	0	1
October, ...	2	120	1	0	0
November, . . .	1	300	0	0	0
December, . . .	0	0	1	0	0
1907 Total, . .	8	975	11	4	9
1906 Total, . .	1			1	0

1908.

Month.	Producing Wells Completed.	New Production	Gas.	Dry.	Abandoned
January, ...	3	210	1	0	0
February, . . .	2	580	1	0	0
March, ...	1	80	0	0	0
April, ...	5	535	0	0	0
May, ...	4	1,570	2	0	0
June, ...	3	1,200	0	1	2
July, ...	1	140	0	0	0
August, ...	6	1,005	1	1	0
September, . . .	5	1,425	0	0	0
October, ...	5	4,395	1	3	0
November, . . .	3	190	1	2	0
December, . . .	5	3,025	1	0	0
1908 Total, .	43	14,355	8	7	2

1909.

Month.	Producing Wells Completed.	New Production	Gas.	Dry.	Abandoned
January, ...	5	550	0	1	0
February, . . .	4	465	3	3	1
March, ...	9	1,045	2	2	4
April, ...	5	500	0	5	2
May, ...	8	535	2	3	0
June, ...	8	760	1	2	0
July, ...	8	875	0	11	0
August, ...	7	515	0	1	3
September, . . .	3	590	5	2	0
October, ...	3	140	1	1	0
November, . . .	5	2,605	3	0	0
December, . . .	4	170	2	2	0
	69	8,750	19	33	10

[a] Compiled from the *Oil Investors' Journal.*

III. Geology.

In drilling for oil and gas in the Caddo field three systems of rocks are met with: the Quaternary, Tertiary, and Cretaceous. The Quaternary, made up of red clays and sands, is found on the surface. The Tertiary, composed of clays and sands with large boulders or concretions, underlies the Quaternary. The Cretaceous system, the oldest, is made up of more consolidated rocks, "gumbo," shale, chalk, etc.

The beds all dip towards the south, and the Cretaceous formations encountered in the Caddo field outcrop in Arkansas and Texas. Several structural peculiarities exist in Louisiana, the most noted being the Sabine uplift, due probably to a great crustal movement. Of this, Prof. G. D. Harris[1] says:

> "It is the great Sabine uplift that affords the proper structure for the collection and retention of great quantities of oil and gas in northwest Louisiana. The boundary of this uplift to the north, in the vicinity of Vivian, ... is clearly defined. Likewise to the south, from near Natchitoches to the Sabine. So, too, there is apparently a high dip to the east, in the Cretaceous beds along the course of the Red River. But to the west we have no sure proof that the Sabine uplift . . . . may not be continuous with . . . . [an] area further west in Texas. In other words, the insular mass . . . . [the Sabine uplift] might perhaps be more properly shown in the form of a peninsula connecting with the main land toward the west. . . . . Caddo field is near the north angle of the Sabine uplift. Its oil and gas evidently come up the north and east slopes of this uplift and then become entrapped beneath Upper Cretaceous and Eocene impervious beds. But the final distribution of these substances over the field is seemingly due to secondary structural features, slight anticlines and difference of porosity of the rocks in which the oil is contained."

Gas in large quantities seems to have, in the Caddo field, a more general distribution than oil.

There are apparently four horizons in the Caddo field at which oil and gas occur. These all belong to the Cretaceous system. Table II. presents a generalized section of the Caddo field, with a few selected logs.

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[1] *Bulletin No. 8, Louisiana Geological Survey*, p. 6 (1909).

TABLE II.—*Generalized Section of the Caddo Oil-Field.*

System.	Series.	Stage.	Kind of Material	Approx Thickness.	Approx. Depth
				Ft.	Ft.
Quaternary.			Red and gray sand, clay, gravel..........	20	20
Tertiary.	Eocene.	Sabine.	Dark lignitic sands and clays, with calcareous boulders....	430	450
		Midway.	Dark clay, with occasionally a limestone bed....	200	650
Cretaceous (Upper.)	Montana.	Arkadelphia.	Dark stiff clay...............	150	800
		Nacatoch.	"Shreveport" or "Caddo" gas-sand; contains some hard layers...............	130	930
	Colorado.	Marlbrook.	Blue marl with chalky layers about 1,150 ft. (Saratoga chalk)..........	350	1,280
		Austin.	Chalky layers with many fossil fragments, often with strong odor of oil. Occasionally good oil about 1,575 ft. Gas common. The so-called "Annona" chalk......................	320	1,600
			Chalk, clay, and sand, with hard pyrite layers. The Brownstown beds......	200	1,800
		Eagle Ford.	Blossom sands. Gas at 1,800 ft....................	50	1,850
			Blue tough clays with hard limestone and pyrite layers. The "Eagle Ford clays."...	350	2,200
	Dakota.	Woodbine.	These sand-beds, reached at depths ranging from 2,140 to 2,300 ft., according to local structural features, contain the "deep" oil and gas of the Caddo fields. Salt water is common. Thickness unknown, (100 ft.)		

LOG OF FILER No. 1.

Record received from Mr. Plumb.

	Thickness	Depth
	Ft.	Ft
Clay	75	75
Sand	5	80
Dark clay	20	100
Rock	4	104
Clay	11	115
Rock	3	118
Shale	57	175
Rock	2	177
Shale	148	325
Rock	3	328
Shale	172	500
Rock	4	504
Gumbo	121	625
Rock	5	630
Gumbo	50	680
Shale, hard	20	700
Shale, soft—some gas	20	720
Shale and gumbo,	90	810
Rock	80	890
Shale, hard	60	950
Gumbo	40	990
Shale	50	1,040
Gumbo	29	1,069
Gumbo	51	1,120
Shale	80	1,200
Lime, soft	100	1,300
Lime, hard	140	1,440
Rock	10	1,450
Lime, soft	80	1,530
Rock	10	1,540
Lime, soft	25	1,565
Shale	115	1,680
Gumbo	80	1,760
Shale, hard	115	1,875
Rock	5	1,880
Shale	110	1,990
Rock	8	1,998
Shale	52	2,050
Rock	10	2,060
Shale	75	2,135
Rock	5	2,140
Shale	100	2,240
Good oil-sand	12	2,252

LOG OF C. G. O. HOSTETTER No. 1 GAS.

Record received from B. G. Dawes.

	Thickness	Depth.
	Ft.	Ft
Brown and yellow clay	15	15
Gumbo and hard blue shale	11	26
Gray colored lime shale	2	28
Soft blue shale	24	52
Gray colored hard lime shell	1	53
Gumbo and soft blue shale	27	80
Gray hard lime shell	2	82
Soft blue shale	43	125
Gray hard lime shell	3	128
Gumbo, hard and stiff	32	160
Rock shell, sandy and soft	2	162
Gumbo, blue and hard	58	220
Sand rock, with gas	2	222
Stiff blue shale	58	280
Soft sand rock	3	283
Sandy brown shale, with gas	27	310
Hard sand rock	1	311
Blue hard gumbo	64	375
Soft sand rock shell	2	377
Gumbo	45	422
Hard sand rock	3	425
Hard tough gumbo	65	490
Soft sand rock	3	493
Gumbo and light blue shale	144	637
Kaolin, white shale, or gypsum,	11	648
Soft sand rock (2,500,000 to 3,000,000 "gasser")	9	657
Kaolin or white shale, sandy at base	10	667
Hard stiff gumbo	91	758
Hard sand rock	3	761
Gumbo with sandy shale at base,	16	777
Hard sandy shale	3	780
Caddo gas-sand, quite hard, light in color, very sharp and gritty, with some gas	3.5	783.5

LOG OF HEYWOOD No. 1, FEE.

Record received from H. H. Jones, Driller.

	Thickness. Ft	Depth Ft
Clay, variegated soft	80	80
Bluish sand, water-bearing	10	90
Clay, dark, soft	50	140
Hard dark rock	2	142
Dark soft clay	8	150
Hard dark rock	3	153
Gumbo and shale, dark, full of boulders	447	600
Gumbo and dark shales	110	710
Shale, dark, with persistent gas	10	720
Gumbo, dark and very tough	100	820
Rough, lime-like rock with gas	10	830
Dry sand rock, with gas and oil smell	34	864
Rough lime rock	5	869
Dry sand rock, with oil and gas	61	930
Hard rock	2	932
Sand rock	8	940
(Gas occurred in this well from 820 to 940 ft.)		
Shales, with streaks of gumbo	39	979
Rock	2	981
Shale and gumbo with hard streaks	163	1,144
Chark, white and soft	11	1,155
Dark gumbo	53	1,208
Dark shale	21	1,229
Dark gumbo	43	1,272
Chalk, clayey to white, oil from 1,470 to 1,520 ft.	328	1,600
Black rocky gumbo	100	1,700
Black sandy gumbo	80	1,780
Rock with some pyrite	5	1,785
Coarse, variegated sand with oil smell	13	1,798
Rock, in streaks and some pyrite	62	1,860
Gumbo and shale	112	1,972
Tough gumbo	40	2,012
Shale and dark gumbo	118	2,130
Rocky, with pyrite and shells	13	2,143
Red and dark shale	37	2,180
Blue shale	16	2,196
Rock	2	2,198
Soft rock	2	2,200
Hard shale	24	2,224
Oil shale	3	2,227
Hard shale	5	2,232
Oil shale	4	2,236
Hard shale	3	2,239
Rock	3	2,242
Brittle rock	10	2,252
Shale	5	2,257

The Caddo oil is generally black, and ranges in specific gravity from 21.3° to 41° Baumé. The oil carries much water, but no sulphur.

An analysis of Caddo gas, made by Prof. C. F. Phillips, gave:

	Per Cent.
Methane,	95.00
Nitrogen,	2 56
Carbon dioxide,	2.34
Hydrogen,	0.00
Carbon monoxide,	0 00
Ethylene,	0.00
Sulphide (hydrogen sulphide?),	0.01

IV. Topography.

The Caddo field lies in the Gulf Coastal Plain—an area of low and rounded relief rarely exceeding 500 ft. in elevation—and contains several small lakes, which are the most important recent topographic features of NW. Louisiana. Fig. 3 is a view of Ferry lake, near the center of the field, and Fig. 4 shows a small salt lake of the Producers' No. 2 blow-out. These lakes, due to the damming up of the Red river, have been formed since the fifteenth century. However, they are now returning to their former level as tributary streams.

Over most of the Caddo field are found low, circular, mound-like elevations, Fig. 5, from 20 to 100 ft. in diameter, with a maximum elevation of 6 ft. These mounds are very noticeable throughout the field, because of their persistence and wide distribution.

The condition of the ground and roads is far from good. The lakes and bayous almost completely surround with water the whole central portion of the field, and make the hauling of pipe and machinery exceedingly hard work. Salt-water ponds and swampy patches occur here and there; and, in general, the surface throughout the whole field presents swampy conditions. Fig. 6 shows the effect of a salt-water spray on surrounding trees and land.

V. Ownership of Land.

An operator may either own the land upon which he drills or lease it. Probably in the early development most drilling was done upon owned land. As the development increased, however, outside capital came in, to operate mostly on leased land.

In 1907, two years after the opening of the field, land could be bought at from $25 to $50 per acre. At the time of the great boom in December, 1908, the price jumped to $500 and $1,000. At present the cost per acre leased is from $50 to $500 per annum, with one-eighth of the product as the usual royalty.

In January, 1910, the assessor of Caddo parish announced that oil-lands would be assessed on the following basis: The owner of the fee to be assessed on the value of the land from a surface stand-point. The oil-rights to be assessed at one-eighth (or the royalty) to the owner of the fee, and seven-eighths to the lessee.

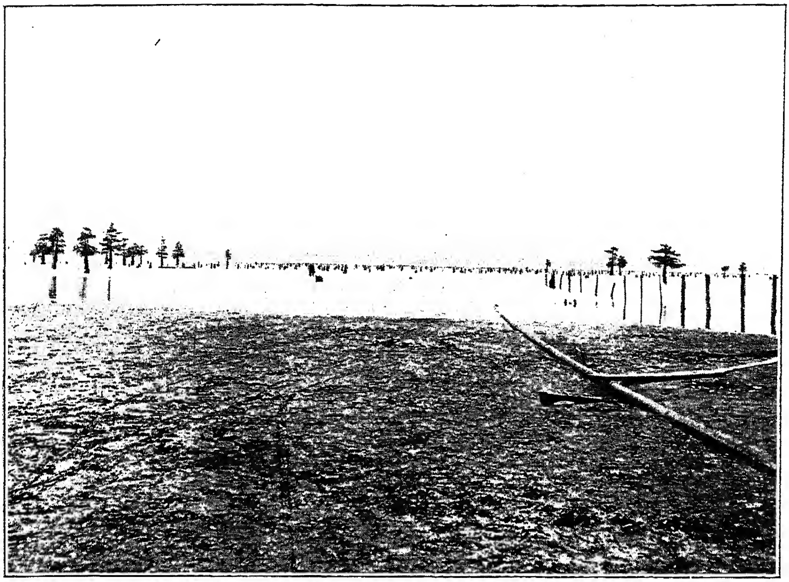
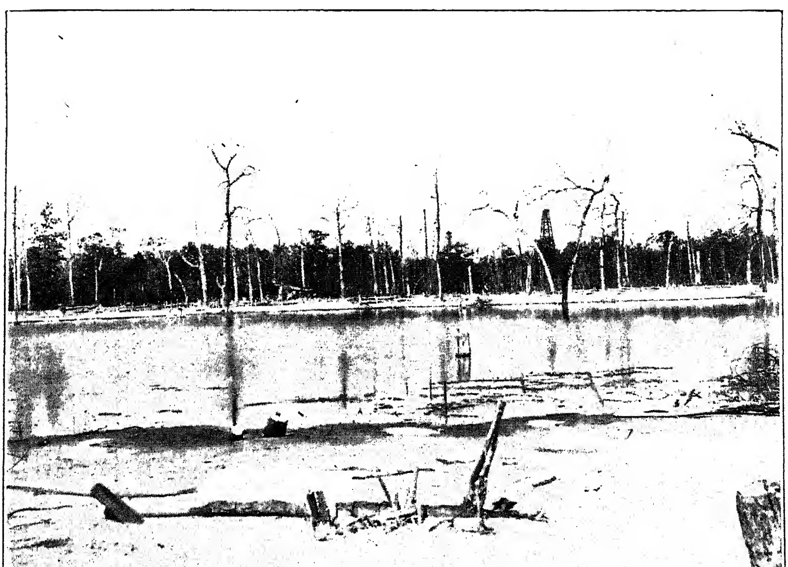


FIG. 3.—VIEW SOUTHWEST OVER FERRY LAKE NEAR CENTER OF FIELD.



From *Bulletin No.* 429, *U. S. Geological Survey* (1910).

FIG. 4.—SALT LAKE MARKING LOCATION OF THE PRODUCERS' NO. 2 BLOW-OUT.

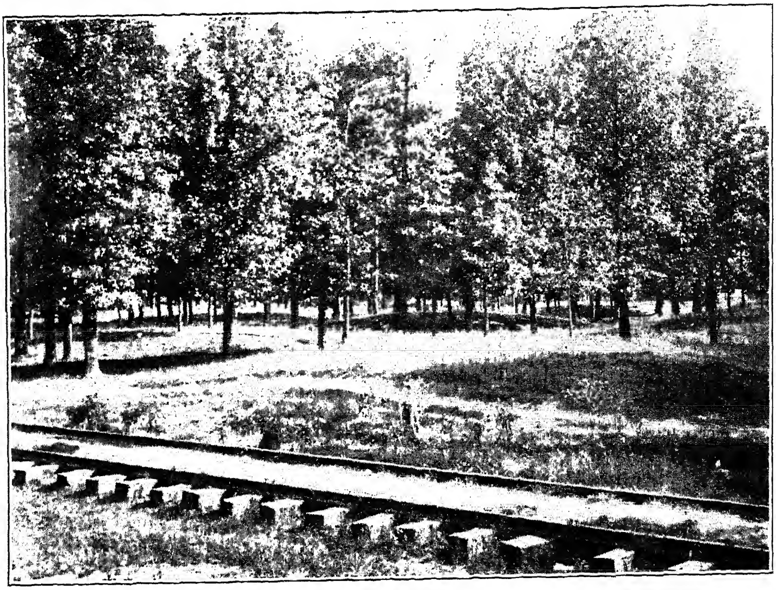


FIG. 5.—GAS-MOUNDS NORTHWEST OF TEXARKANA, TEX., A NOTICEABLE FEATURE OF THE TOPOGRAPHY OF THE CADDO FIELD.

FIG. 6.—PRODUCERS' NO. 1 SALT-WATER GUSHER, SHOWING THE EFFECT OF THIS SALT-WATER SPRAY UPON THE TREES AND THE SURROUNDING LAND.

VI. Drilling-Operations.

1. *Locating and Erecting Derrick.*

It is the common practice to place the derrick on one of the so-called gas-mounds, or as near as possible to a producing well. The average hire of a surveyor to locate a well is from $10 to $25.

The rotary derrick used can be easily framed and erected by any competent carpenter, although there are rig-builders here, as in every field. The building of a rig is by no means easy work; and every man who works up in the rig as it is erected generally receives $5 per day. The derrick is usually 84 ft. high to the top of the crown-block, with a 22-ft. base. The cost of a rig or derrick, including lumber and labor, is about $250.

2. *Machinery Equipment.*

Rigging up, one of the most expensive items to the operator or contractor, includes the purchasing and transporting of the rotary, engine, hoisting-machine, pumps, boiler, casing, and pipe. The cost of transportation of machinery and pipe from the railroad to the well varies from $150 to $200, according to distance.

There are several good rotaries now in use in the various oil-fields, each one possessing advantages for a particular field. I might mention the Parker rotary and the Oil-Well Supply rotary as two types used in the Caddo field. These rotaries vary in weight from 900 to 4,000 lb., and in size of pipe handled from 1.5 to 20 in. The price varies from $225 to $1,600.

Several makes of engines of 20, 23, 25 and 28 h-p. are in use. The price of one of these engines complete varies from $320 to $365.

The hoisting-machines vary in weight from 1,990 to 2,800 lb., and in price from $180 to $250.

Pumps of several styles are used in connection with the hydraulic rotary system. These are the Smith-Vaile, Knowles, Special Duplex, Gardner, and Parker rotary drilling-pumps. The price varies from $220 to $510.

The boiler, placed from 100 to 200 ft. away from the well,

to insure against danger from an unexpected flow of gas or oil, is run by gas, generally supplied by a pipe-line from some gas-well in the field. The oil-country boilers, of locomotive type, are generally used. These are made of open-hearth flange steel, having a tensile strength of 60,000 lb. per square inch, and an elastic limit of 30,000 lb. They are tested at 160 lb. hydrostatic pressure and 125 lb. steam pressure per square inch.

The total cost for a complete rotary outfit, including tools and materials, f. o. b., is estimated at $2,825, not including freight-charges. The weight of this outfit is about 30,000 pounds.

3. *Methods of Drilling.*

The hydraulic rotary method of drilling, used exclusively in the Caddo field, is a modification of the Fauvell system, invented in 1845, and used for some time in several of the European oil-fields. It is rapid and economical, when used in unconsolidated formations, such as clay, sand, or quicksand.

Several styles of rotary are used, but all work on the same principle. The principal features of the rotary outfit are the derrick, the rotary machine, the hoisting-gear, the engine, the boiler, and the pump. In addition are the numerous accessories and appliances for connecting the machinery; for attaching the pump to the boring-tube or casing; for hoisting and handling the tubes or casing; the bits or cutting-shoes; the pipe and casing.

The rotary method consists in rapidly turning a column of pipe, the lower end of which is armed with a steel shoe having a serrated edge or a bit for cutting. The drill-pipe is held by a chuck, and the descent of the pipe is controlled by the driller by means of a feeding-device. Water is kept under constant high pressure in the pipe and the cuttings are thus forced to the surface, passing up on the outside of the pipe with the water. This ascending current of water keeps the hole clean and allows the drill to turn freely. It is essential that the flow of water should be continuous, and a drilling-outfit is always supplied with two force-pumps in order to prevent any stoppage of the flow. If the drill has passed through a pervious stratum, such as a bed of loose sand, the ascending water is liable to pass into that stratum instead of returning to the

surface. This quickly results in the clogging of the hole; and in order to prevent it the water which is pumped in is mixed with a large amount of fine clay. By this means the outlets through porous beds are sealed up, the unconsolidated material forming the walls of the hole is prevented from caving, and the water returns unimpeded to the surface. When drilling through quicksand or similar formations, a back-pressure valve is inserted in the coupling above the first joint of pipe, to prevent water and sand from rushing back into the pipe when the service is disconnected. This also assists materially in sustaining the wall of the well and permitting the pipe to turn. When the pipe has reached the desired depth, the valve, which is constructed of brittle material, is punched out, and the pieces either removed or forced down outside of the pipe.

When the formation is not of sufficient hardness to form a core, but washes out as the pipe advances, the rotary steel shoe with a serrated edge is used. In cases where the formation is composed of clay of sufficient density to retain the wall of the well in place, a smaller pipe is used, armed with a perforated bit, through which the water is forced from above, and when the depth is reached where it may be desirable to insert the casing, the drill-pipe is removed and the casing is inserted in the hole thus prepared for it.

Two forms of bit are generally used, the fish-tail and the core-barrel. The fish-tail bit requires considerable care both in making and in dressing. It cuts through comparatively hard formations, and must be so shaped that, as the points advance and outline the hole, the center portion crumbles the core which tends to form, and the small holes must direct the jets of water in such a way that the bit as well as the hole is kept clean. The core-barrel bit is used when a stratum is encountered which is too hard to yield readily to the fish-tail bit. This core-barrel consists of a piece of steel pipe, about 8 ft. long, swaged at the upper end to connect to the drill-pipe. The lower end is left smooth, and two or three holes are drilled a short distance above, to permit the water to pass out and return to the surface. Chilled iron or steel particles averaging about the size of bird-shot are fed down the inside of the pipe in small quantities and find their way to the bottom of the hole, where they are rolled between the bottom of the core and the

rock, rapidly crushing the latter. As the cuttings rise above the heavier shot, they are caught in the current of water and carried to the surface. The core is removed by means of an extractor consisting of a piece of pipe of the same size as the core-barrel and provided with short inwardly-projecting steel springs which engage the core and carry it up when the pipe is removed. It is necessary to exercise care, in pumping water, to admit only enough to remove the cuttings and not wash out the shot.

In starting a well arrangements are generally made first for a 12-in. casing; and in boring for this size the drill-pipe is generally made of 6-in. casing with a 13.5-in bit. This size of bit is used to allow the collars at the joints of the 12-in. casing to slide past without damaging the wall of the well. The length of 12-in. casing used varies from 500 to 800 ft., depending largely upon the nature of the ground and the skill of the driller. The hole is generally left open until the whole depth calculated for one string of casing has been drilled. This generally extends until a hard stratum is met, upon which the casing to this depth may stand.

Following the 12-in. casing, the hole is next drilled for an 8-in. or 9-in. casing. In either case a 10.5-in. bit is used. After the 9-in., a 6-in. casing is used; then a 4-in., and then finally a 2-in. casing.

The drilling depends largely upon the driller, who controls the engine and the drilling-pipe. It is necessary to keep the hole clean, and the pumps are generally kept going even though not drilling. Very often the well becomes clogged and it is necessary to pull out the whole string of pipe in the hole. The bits also require dressing, which necessitates the withdrawal of the pipe. The pipe is generally drawn out in sections of three lengths, or about 60 ft. It is stood up on the derrick-floor until put in the hole again. The cuttings as they are washed out by the water under the rotary are examined every 20 to 50 ft., and a log or record of the well is kept by the driller.

When the oil-bearing formation is reached, the oil will be noticeable on the water as it passes into the slush-pit. When, in the driller's judgment, the drill has penetrated the oil-bearing formation to a sufficient depth to assure a good flow, the pipe is removed and the well is bailed. A strainer is generally

set at the bottom of the hole before bailing. The 10- or 20-ft. bailer, generally used, is lowered to the bottom of the well, the dart-valve opened, and the water allowed to fill the bailer. It is then pulled up and the water dumped into the slush-pit. Before the well is entirely bailed a gate-valve is fitted to the casing, in such a way as to permit a rapid closing of the well if desired. As soon as enough water has been removed to reduce the pressure the oil rises and the well flows. The well is allowed to clean itself of all loose pieces of rock or gravel, when the valve is closed and the well shut in. The well is then connected by a pipe-line to storage-tanks, a cooking-tank, or the loading-rack. In drilling for gas great caution must be taken to prevent a blow-out. A special quick-closing gate-valve set in cement is now used.

Five men usually make up the crew of a well: the driller, derrick-man, boiler-man, and two helpers. The driller is in complete charge of the drilling of the well. The daily wage of the driller varies from $5 to $6. Night-drillers sometimes receive more than this. The derrick-man receives from $3 to $5 per day or night, and the helpers $3 per day or night.

The average length of time required to drill a deep well, that is, 2,200 ft., in the Caddo field is from 120 to 180 days. Very often a night- and a day-crew are employed and the length of time is then reduced to from 60 to 90 days. The length of time required to drill a shallow well (800 ft.) is 30 days, or about 18 days, drilling day and night.

Besides the hydraulic rotary system of drilling, the standard or cable-tools method has been used in the Caddo field, but without satisfactory success. J. C. McCue, superintendent of the Producers' Oil Co., writes: "You will please note that I answer by saying—'Impossible to drill with cable tools.' We find that the formation is such that cable tools cannot be used."

The drilling of a well by this cable-tools method is done with a steel drill, measuring with its fittings 30 ft. in length, and weighing from 1 to 1.5 tons. This drill is continually lifted and dropped in the hole, the force of its impact breaking the rock. At intervals the débris is removed by a sand-pump—a tube with a valve at the bottom, which is lowered into the hole and drawn out, bringing the cuttings with it.

Four men are required on a standard rig—the driller, boiler-

man, and two helpers. The wages of the driller are $180 per month and of the helpers $4 per day. The total cost to build a standard rig is about $750, while the estimated cost of tools, not including machinery, is about $2,000.

Very often the oil in a well is not sufficient for the well to flow, in which case a pump is put on the well. In the Caddo field few gushers are brought in; hence a large majority of the wells are pumped. A standard rig, built for the pumping, costs about $600. Recently two air-compressors have been installed. Wells producing large quantities of water with the oil are best blown by compressed air; indeed, it is the only way to handle them. Fig. 7 is a view of a well on Pine Island blown by an air-compressor.

Frequently, when the oil-bearing formation is reached, the oil either does not flow at all or flows only in small quantities. Instead of putting the well on the pump, it is sometimes "shot." By exploding a charge of nitro-glycerine at the bottom of the hole, the surrounding rock is broken up and the flow of oil stimulated. The shooter lowers the glycerine into the well in cylindrical tin shells. The well is then filled for a couple of hundred feet with water to "tamp" the charge. The shooter lights the fuse of a "jack squib," a long slender shell filled with a small charge of glycerine, a fulminating-cap, and a slow-burning fuse, and starts the squib down the well. I have record of only one well having been shot in the Caddo field, and this with no success. The explosive in this case cost $275.

VII. Treatment of Product.

1. *Cooking.*

Several of the wells in the Caddo field produce oil containing much mud and water. Since the maximum percentage of dirt allowable is 3 per cent., it is necessary to "cook" the oil to free it from such impurities. The stationary "cooker," commonly used, has a capacity of 1,000 barrels in 24 hr. The cooking-tanks are built as follows: A small gas-mound is selected, in which a hole is excavated as large as the tank is to be, but not deeper than the level of the surface surrounding the mound, Fig. 8. The tank is then built inside the pit. The walls are double, of 2-in. plank, with a space between, which is

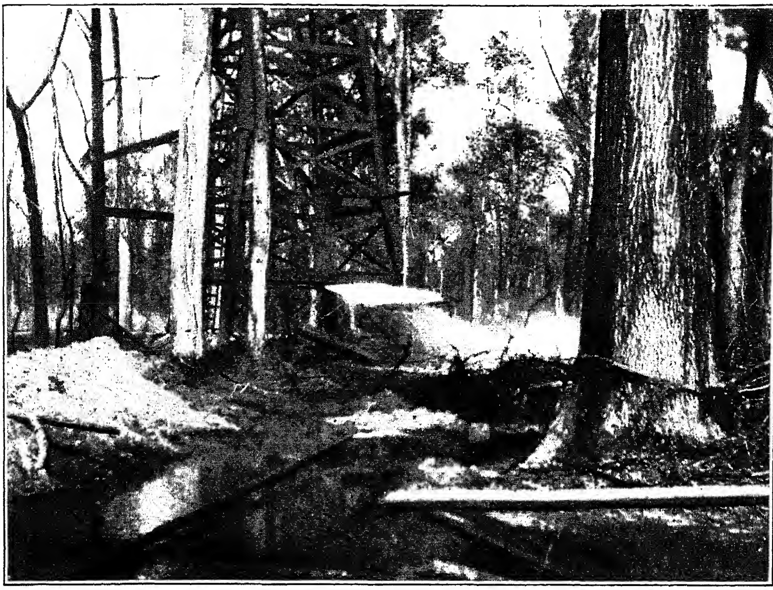
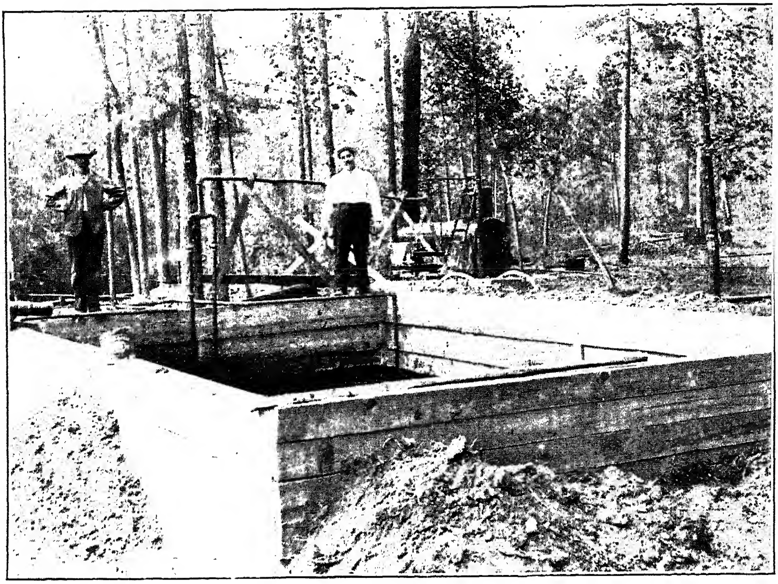


FIG. 7.—WELL ON PINE ISLAND BEING BLOWN BY AN AIR-COMPRESSOR.

FIG. 8.—EXCAVATING A GAS-MOUND FOR THE BUILDING OF A COOKING-TANK.



From *Bulletin No.* 8, *Louisiana Geological Survey* (1909).

FIG. 9.—COOKING-TANK FOR THE BOILING OF OIL CARRYING SALT-WATER AND MUD.

FIG. 10.—WOODEN STORAGE-TANKS, 1,200 BARRELS CAPACITY, OF THE FILER OIL CO.

afterwards filled with sand and the cracks calked with oakum, Fig. 9. Coils of 1.25-in. steam-pipe are then put in and the cooker is ready. In building the tank it is necessary to provide an opening for drawing off the water and letting the oil out after it has been steamed, and still another to let out the mud when the pit is cleaned. In cleaning these pits a hose attached to a water-pump is used, and water is pumped into the pit with great force.

The temperature of the oil is just as hot as the steam can make it. The length of time of cooking varies according to the quality of the oil. If it contains 80 per cent. of mud and water it will require 8 hr. of steaming for 400 barrels of the oil.

2. *Storage.*

Every producing well has one or more storage-tanks for its product. The tanks are of wood, bound with iron hoops, and have a truncated conical shape, and a capacity of 100, 250, 600, 800, 1,000, or 1,200 barrels. In the Caddo field the 600- and 1,200-barrel sizes are generally used, Fig. 10. After the well-connections are made and the pipe-line is connected to the tank, a square wooden house is built over the tank to protect it from the weather. The cost of a 600-barrel wooden tank is about $300, and that of a 1,200-barrel one about $500.

A company operating a number of producing wells in this field generally builds a large steel storage-tank with a capacity of from 30,000 to 50,000 barrels, at a cost of about $0.28 per barrel of capacity, or $9,800 for a 35,000-barrel tank.

Companies operating more than one well employ a gauger, whose duty it is each day to sample and measure the depth of the oil in the storage-tanks. The wages of a gauger are $3.50 per day.

3. *Transportation.*

The oil, after it has been cooked, or direct from the well, is piped to the loading-rack. In the Caddo field there are at present five loading-racks, which vary in capacity from 20 to 60 cars. From there the oil goes direct to the refinery or to the buyer over the Kansas City Southern railway.

Car-shipments of oil from the Caddo field (barrels of 42 gal.):

Year 1909.	Oil City.	Mooringsport.	Vivian.	Lewis.	Total
January,	51,400	6,359	......	......	57,759
February,	81,690	6,699	......	......	88,389
March,	55,925	10,145	.....	......	66,070
April,	56,622	3,658	....	2,935	63,215
May,	7,979	....	....	2,086	40,065
June,	51,529	18,430	4,447	3,115	77,521
July,	48,756	35,593	4,025	2,053	90,427
August,	38,162	23,022	1,646	2,380	65,210
September,	37,414	26,633	....	2,245	63,292
October,	49,313	41,922	....	1,076	92,311
November,	49,852	15,737	....	....	65,589
December,	49,599	11,380	......	....	60,979
Totals,	608,241	196,578	10,118	15,890	830,827

Runs of crude oil from the Caddo field:

Year.	Barrels.
1905,	4,650
1906,	....
1907,	48,266
1908,	499,937
1909,	985,226
Total,	1,538,079

The gas-wells are connected with the main line, or the product is used to supply the drilling wells in the field.

VIII. Consumption of Oil and Gas.

A large part of the crude oil which does not go to the refinery is used by the Texas and Louisiana railroads, consuming daily about 16,500 barrels of fuel-oil. Table III. gives the approximate quantity of oil used by the various railroads in 1909.

TABLE III.—*Consumption of Crude Oil by Railroads in* 1909.[a]

Name.	Barrels
Southern Pacific R. R.,	2,195,000
Gulf, Colorado & Sante Fe R. R,	1,793,288
Kansas City Southern R R,	665,000
San Antonio & Aransas Pass R. R.,	480,000
International & Great Northern R. R.,	360,000
St. Louis, Brownsville & Mexico R R.,	145,000
Trinity & Brazos Valley R. R.,	145,000
Galveston, Houston & Henderson R. R.,	60,000
Gulf & Interstate R. R,	48,000
Texas & Pacific R R,	48,000
Houston Belt & Terminal R. R.,	48,000
Galveston Wharf & Terminal R. R.,	36,000
Total,	6,023,288
Daily average,	16,502

[a] From the *Oil Investors' Journal*, vol. viii., No. 18, p. 85 (Feb. 20, 1910).

All but a small portion of the oil consumed by the railroads named in Table III. is burned on the locomotives. A small quantity is used by several of the lines at their shops and for pumping water.

Table III. shows the total number of barrels of oil used by the Kansas City Southern railway. The amount of crude oil supplied to this railway from the Caddo field is as follows: From Oil City, 257,461; from Vivian, La., 18,412; and from Lewis, La., 1,087, making a total of 276,960 barrels.

Caddo gas is used in Shreveport, La., 20 miles south of the field. The price charged is $0.30 per 1,000 cu. ft. for domestic purposes and about $0.15 for manufacturing uses.

A flat rate of $0.50 per month for heating and $0.10 per month for lighting is maintained in the field.

Caddo gas is also used in Texarkana, Ark.; Dixie and Belcher, La. With the completion of the new lines, Caddo gas will be carried to St. Louis, Mo., to Houston, Tex., and possibly to New Orleans, La.

During the early part of 1909, the estimated daily waste of gas in the field by the two burning gas-wells, the Caddo Gas and Oil Gilbert No. 1, and the C. G. Dawes Trustee No. 1, was about 75,000,000 cubic feet.

IX. Cost.

The general impression is that it costs from $10,000 to $12,000 to drill a deep well in Caddo field.

C. W. Brown, of Brown Brothers, well-contractors, has stated the cost of drilling an oil-well to the depth of 2,200 ft. at approximately $10,000, and that of a gas well 1,000 ft. deep at about $4,000.

The following data show the average cost of drilling a deep well.

Cost to lease one acre of land,	$300
Total cost of rig or derrick,	250
Cost of machinery, tools, and material,	2,824
Cost of transportation of above to well,	100
Cost of driller for 150 days at $5.00 per day,	750
Cost of four helpers for 150 days at $3.00 per day,	1,800
Cost of pipe and casing,	3,265
Total,	$9,289

With oil worth $0.44 a barrel, it would be necessary to have a production of 21,111 barrels in order to pay expenses. When one considers that a good well in the Caddo field means 200 barrels per day, some idea of the returns an operator receives can be obtained.

The review of the Caddo field for 1909 shows 69 oil-wells completed, with an initial output of 8,750, and an average of 127 barrels; 19 "gassers" completed; 33 dry holes drilled; and 10 wells abandoned.

The 69 oil-wells represent a total expenditure of about $690,000. The 19 gas-wells represent about $76,000. Assuming 25 of the dry holes to have been deep holes, we have $250,000 spent on these holes; and assuming 8 to have been "gassers," we have $32,000 for these. The total expenditures for oil-wells would thus be $940,000 and for gas-wells $108,000. No doubt the same machinery was used to drill several wells; but still the foregoing figures furnish an idea of the amount of money spent in the field during the year 1909. Besides the drilling, there are the pipe-lines, storage-tanks, pumps, loading-racks, and cooking for the oil; also the pipe-lines for the gas.

The cost of lap-welded pipe is, at the lowest average, about 3 cents per pound, including the joint. The cost of hauling, lay-

ing, and painting, per foot, as given by the *Oil Investors' Journal*, is:

Size	Cost.	Hauling	Laying.	Painting.	Total
2-in.	$0.24	$0.01	$0.03	$0.01	$0.29
4-in.	0.41	0.02	0.03	0.01	0.47
6-in	0.65	0.02	0.04	0.02	0.73
8-in.	1.05	0.02	0.04	0.03	1.13
10-in.	1.30	0.03	0.05	0.03	1.41
12-in.	1.70	0.04	0.05	0.03	1.82
16-in.	2 80	0.05	0.06	0.04	2 95
18-in	4.00	0.06	0.07	0.04	4.17
24-in.	5.10	0.08	0.08	0.05	5 31
30-in.	7.50	0.09	0.09	0 06	7.74

These prices, however, are sometimes far from the actual cost.

Let us consider the proposed gas-lines to New Orleans and St. Louis, the length of each being about 400 miles. The former, a great undertaking, will necessitate the laying of 18-in. pipe for 400 miles through swamps and bayous, and the crossing of the Atchafalaya and the Mississippi rivers. Assuming $4.15 as an average total cost per foot, the cost will amount to $8,764,800. Very few appreciate the enormous expenditures represented by these pipe-lines.

Another considerable cost to a company operating on a large scale are the large steel storage- and cooking-tanks. The Producers' Oil Co. has a 50,000-barrel tank in the Caddo field, which cost about $14,000. The Caddo Gas & Oil Co. owns a 35,000-barrel steel tank, which cost about $9,800.

X. Life of the Field.

Under this head, Prof. G. D. Harris[2] points out the important circumstance that the Caddo field is not one of extreme local concentration, and has nothing in common with the Beaumont or Jennings fields. In everything except its geologic age, it resembles the oil-fields of western Pennsylvania, West Virginia, Ohio, and Illinois. Its vastness speaks well for both quantity of oil and gas and the longevity of the field as a productive source of these materials.

I wish to express publicly my thanks to Prof. Heinrich Ries for criticism and suggestions; to Prof. G. D. Harris for suggestions and for the photographs used in this article, and to Donald Steel, of Cornell University, for helpful criticisms.

[2] *Oil Investors' Journal*, vol. vii., No. 11, p. 18 (Nov. 6, 1908).

Tunnel-Driving in the Alps.

BY W. L. SAUNDERS, NEW YORK, N. Y.

(Wilkes-Barre Meeting, June, 1911.)

I. INTRODUCTION.

IT is now generally admitted by experts that at least so far as rapid progress is concerned the Alpine system of tunnel-driving is superior to any other. This is perhaps natural in view of the record of experience in driving tunnels through the Alps. These great mountain-chains cannot be treated in the ordinary way by shaft-sinking and driving headings, thus multiplying the points of attack. The work must be done from the ends only, hence speed in driving is of the utmost importance; and, as necessity is the mother of invention, the concentration of effort to make progress has resulted in an organization and in mechanical appliances that have produced results so much in excess of the usual practice, even in America, that a discussion of the subject in detail should be of much value to the engineer and contractor.

The first Alpine tunnel was the Mont Cenis, length 7.5 miles, driven with a progress that averaged about 7.75 ft. per day. Next came the Saint Gothard, 9.5 miles long, 18 ft. per day; Arlberg, 6.5 miles long, 27.25 ft. per day; Simplon, 12.25 miles long, 36 ft. per day.

The figures represent progress when driving from two headings, so that by dividing them in two we get the daily single-heading progress. The latest of the Alpine tunnels is the Loetschberg, now being driven. In this work the world's record has been beaten by a single day's record in one heading of 36 ft. and by an average daily record in one heading of 29.5 feet.

The Alpine range forms a natural barrier between a large section of northern and southern Europe. This range extends from the southeast of France to the frontier of Hungary, between Italy and the plains of southern Germany. The contour

of the Alpine range is such that a traveler proceeding from Italy to France, Switzerland, or Germany, after traversing a comparatively easy pass over the main chain, may be brought abruptly against a second and loftier pass, or he may be compelled to follow a circuitous route which may double the length of the journey. The central portions of these mountains consist of gneiss, schists, granite, and other crystalline rocks. The range is an instructive example where a great mountain-chain has been formed by repeated movements during prolonged geological periods. Archibald Geikie, F.R.S., in describing the geology of this region, says:

"When the paroxysm of elevation had ended, one or more large lakes formed along the northern base of the mountains. In these hollows the Swiss molasse accumulated to a depth of more than 6,000 ft.—a great pile of slowly formed gravels, sands and clays. That the sea gained occasionally access to the region is shown by the interpolation of bands containing marine organisms Not improbably a gradual subsidence of the region was going on during the formation of the molasse. But during the Miocene period another great epoch of mountain making was ushered in. The lakes disappeared, and their thick sediments were thrust up into large, broken, mountain masses. Since that great movement no paroxysm seems to have affected the Alpine region."

Before surveys had been made of this mountain barrier the needs of war and commerce actuated people living on the opposite sides to seek the easiest and most direct route for crossing it, hence as far back as history records we find mention made of passes over the Alps. A pass is usually understood to be a depression between adjacent mountains, over which a trail is made. The Romans, beginning with Julius Cæsar, became acquainted with the easiest and most serviceable passes of the Alps. It was always a difficult matter to cross over these passes, especially in winter, when storms and snow obstructed progress. To cross the Alps with an army, even in ancient times, without modern artillery, was a feat that compared favorably with winning a great battle. Hannibal, 200 years before Christ, crossed over the pass of Little Saint Bernard with an army of 20,000 infantry, 6,000 cavalry, and 37 elephants. This passage took 15 days and one-third of his army was lost *en route*. We have the authority of Livy for the statement that Hannibal, in order to widen some of the passages through the mountains, flaked off the material by heating the rocks and cooling them suddenly with liquids.

The credit for the modern revival of tunnel-construction on a large scale is due to Anne of Lusignan, who in the year 1450 gave it its first impulse by commencing the construction of a tunnel in the Alps, between Nice and Genoa, through the Col di Tenda (height of the pass, 1,800 m.). The work was stopped, but was subsequently continued by Victor Amadeus III. in 1782, finally being abandoned in 1794 in consequence of the invasion of the French; at this time about 2,500 m. of the tunnel is said to have been completed. While the tunnel projected under the Col di Tenda may be regarded as the modern revival of tunneling, there were equally ambitious projects carried out as early as the beginning of the Christian era, when, according to Pliny, the Emperor Claudius completed in A.D. 52 a tunnel from Lake Fucinus (Celano) to the river Liris. This tunnel undertaking is noteworthy as giving some information of the methods, labor, and tools employed in what was considered the greatest public work of the time. This was then by far the largest tunnel in the world, being 3.5 miles long, with a cross-section varying from 10 by 6 ft. to 20 by 9 ft. Forty shafts, some of which were 400 ft. deep, and a number of *cuniculi*, or inclined galleries, were sunk, and the excavated material was drawn up by windlasses, in copper pails of about 10 gal. capacity. It is reported that 30,000 laborers were employed 11 years in its completion. Where diorite, granite, and other hard stone had to be cut, the work was done by tube-drills and saws supplied with corundum or other hard, gritty material and water, the drills leaving a core of rock exactly like that of the modern core-drill.

"By referring to ancient writers—Pliny, Italiana Vitori, Lapidarium of Marbodus—we find that diamonds formed an important adjunct to the 'hewers of stone' as well as the lapidary. And it is thought by Eastern writers that diamond (shamer) pointed tools were used in the construction of Solomon's temple, where 'there was neither hammer, nor axe, nor any tool of iron heard in the house while it was in building.'" [1]

II. Modern Alpine Tunnels.

The era of tunnel-building in the Alps began with the Mont Cenis in 1857. The greatest of the Alpine tunnels, and the longest railway-tunnel in the world, is the Simplon, of

[1] Drinker, *Tunneling, Explosive Compounds, and Rock Drills*, p. 2 (1878).

which the Loetschberg, now under construction, is but an extension.

Since the opening of the Simplon tunnel, connecting Switzerland with Italy, the necessity of forming a route connecting it with the north and northeast of Europe has been apparent and has resulted in undertaking the construction of the Loetschberg tunnel through the Bernese Oberland. The general location of the Loetschberg tunnel and its rail-connections is, from the Simplon tunnel through the Loetschberg to Frutigen and Spiez on Lake Thun; from the entrance of the Simplon to the end of the Loetschberg may almost be considered as one tunnel-system.

A short review of the greater Swiss tunnels will illustrate the continual improvements in the construction-methods and plant employed, and an analysis of the methods of construction in these great tunnels indicates the cause which has led to the progressive increase in the economy and speed of tunnel-construction.

The advance of the heading or "pilot"—irrespective of whether it be driven top or bottom—is the factor controlling the rate of advance, as, under normal conditions, the enlargement to full size, timbering, and lining, readily keep pace with the advance of the heading. The manner of mucking in the headings and the time required to do it average about the same. The increased average gain in the rate of advance has been concurrent with the improvement in the machinery employed in the headings.

Where a great thickness of rock overlies a tunnel, it is necessary to do the work wholly from the two ends, without intermediate shafts. The problem resolves itself into devising the most expeditious way of excavating and removing the rock. Experience has led to great advances in speed and economy, as will be seen from the particulars of the tunnels through the Alps.

	Length. Miles.	Progress Daily. Linear Yards.	Cost per Linear Yard.
Mont Cenis,	7.5	2.57	£226
Saint Gothard,	9.5	6.01	£148
Arlberg,	6.5	9.07	£108
Simplon,	12.4	12	

In 1857 the first blast was fired in connection with the Mont Cenis work; in 1861 machine-drilling was introduced; and in 1871 the tunnel was opened for traffic.

With the exception of about 300 yd., the tunnel is lined throughout with brick or stone. Little interest now attaches to the method of tunneling adopted at Mont Cenis, as it is, in several respects, obsolete. During the first four years of hand-labor the average progress was not more than 9 in. per day on each side of the Alps, but with compressed-air rock-drills the rate towards the end was five times greater.

In 1872 the Saint Gothard tunnel was commenced, and in 1881 the first locomotive ran through it. Mechanical drills were used from the commencement.

Tunneling was carried on by driving in advance a top-heading about 8 ft. sq., then enlarging this sideways, and finally sinking the excavation to invert-level. Air for working the rock-drills was compressed to 7 atmospheres by turbines of about 2,000 h-p. From 6 to 8 Ferroux drills, making about 180 blows per min., were mounted on a carriage pushed to the point of attack. From 13 to 18 holes were drilled by the machine and its 16 attendants to depths of from 2 ft. 7 in. to 4 ft. 3 in. in from 3 to 5 hr., and the work of charging with dynamite, firing, and clearing away was then done by 22 men in from 3 to 4 hr. The charge per hole averaged 1.75 lb., and after firing, a strong current of compressed air was directed over the face of the excavation. Four sets of holes were, under favorable circumstances, drilled in 24 hr., which rendered attainable, in each heading, a progress of 13 ft. a day in such rock as gneiss.

The driving of the Arlberg tunnel was commenced in 1880, and the work completed in little more than three years. The main heading was driven along the bottom of the tunnel and shafts were opened up from 25 to 70 yd. apart, from which smaller headings were driven right and left. The tunnel was enlarged to its full section at different points simultaneously in lengths of 8 yd., the excavation of each requiring about 20 days, and the masonry 14 days. Ferroux percussion air-drills and Brandt rotary hydraulic drills were used, and the performance of the latter was especially satisfactory. After each blast a fine spray of water was injected, which assisted the ventilation materially. In the Saint Gothard tunnel the discharge of the air-drills was relied on for ventilation. In the Arlberg tunnel more than 8,000 cu. ft. of air per min. was thrown in by

ventilators. In long tunnels the quick transport of materials is of equal importance with rapid drilling and blasting. In the Arlberg, to keep pace with the miners, 900 tons of excavated material had to be removed, and 350 tons of masonry to be introduced, daily, at each end of the tunnels, which necessitated the passage of 450 cars. This traffic was carried on over a length of 3.5 miles on a single track of 27-in. gauge with two sidings. When the locomotives ran into the tunnel the fires were damped down, and as the pressure in the boiler was 15 atmospheres, the stored-up heat in the water furnished the necessary power. The cost per linear yard varied according to the thickness of masonry lining and the distance from the mouth of the tunnel. For the first 100 yd. from the entrance the prices per linear yard were £11 8s. for the lower heading; £7 12s. for the upper one; £30 10s. for the unlined tunnel; £45 for the tunnel with a thin lining of masonry; and £124 5s. with a lining 3 ft. thick at the arch, 4 ft. at the sides, and 2 ft. 8 in. at the invert.

III. The Simplon Tunnel.

In 1893 the Jura-Simplon Railway Co. contracted with Brandt, Brandau & Co., of Winterthur, Switzerland, for the construction of a tunnel from Brigue, on the north side, to Iselle, on the Italian side of the Simplon pass, and 12.4 miles in length. The contract time for construction was 5.5 years.

It is but natural that in this Alpine region, where water is available, the engineers should have planned to make use of this power not only for the purpose of compressing air, but for hoists, and even for use in rock-drill cylinders. Water-power was employed for all purposes at each end of the tunnel. The Simplon tunnel runs up-grade from each end towards its center, hence there is a natural drainage, which saves pumping. Engineers, who, without visiting the Simplon during construction, learned of the great progress made there by the hydraulic process, were inclined to adopt the same methods in work less favorably situated. It is one thing to use the hydraulic system where there is a natural race-way taking care of the discharge of the water, and it is quite another thing to expend the power required to pump this water. At the time the East River tunnels, in New York City, were built by S. Pearson & Son for

the Pennsylvania railroad, serious consideration was given to the hydraulic process, because, as is well known to those familiar with conditions about New York, tunneling under the East river involves rock-work, while Hudson river tunneling is entirely through mud. Mr. Moir, who had charge of the East river tunnels, was impressed by the great progress which had been made at Simplon, where hydraulic machinery was used, but, upon investigation, decided that there was no advantage to be gained by using a system which involved lifting the water which had been used for power purposes; hence these East river tunnels were equipped exclusively with compressed-air machinery.

1. *Power.*—At the Swiss end the river Rhone was dammed, the water collected in reservoirs provided with gates, and carried for about 2 miles in a reinforced-concrete flume to the power-house. The power available here was about 2,200 horse-power.

At the Italian end the power was derived from the Diveria, about 2.5 miles above the works. The power-house contained three Pelton wheels, two of 250 h-p. each, and one of 600 h-p.; the wheels were horizontal and ran at 170 rev. per min. The pressure-pumps were operated by these wheels; the highest pressure available at these pumps was about 120 atmospheres per sq. cm., with a capacity of about 65 gal. per min. at a pressure of 1,175 lb. per sq. in. In the pump-room were also two air-compressors of Ingersoll and Buckhardt types. At the Italian end there was also a supplementary steam-power plant of about 220 horse-power.

The distribution of the power was proportioned about as follows: for high-pressure pumps, 700; for air-compressors, 400; for ventilation, 500; for illumination, 100; for machines in the shops, 100; making a total of 1,800 horse-power.

2. *Ventilation.*—For ventilating the tunnel during construction and afterwards, a permanent plant was installed at each end, the power being taken from the 200-h-p. turbines, at each plant, running at 400 rev. per min., and driving two fans of 12.3 ft. diameter. The air-passages from the ventilator-house bifurcate near the tunnel ends. A door at the angle of the bifurcation permits the closing of either fork of the passage; sail-cloth curtains close the tunnel portals. Air could thus be

circulated in either tunnel as desired, its movement being controlled either by compression or by aspiration.

3. *Rock-Temperature.*—The highest temperature encountered during construction was 55° C., and was encountered during 1902, about 8 km. from the Swiss end. The method of refrigerating the air in the workings was the same at each end; cold water was forced into the headings and there broken into spray.

4. *Illumination.*—For illumination at the north end gas was used; at the south end each miner carried an oil-lamp.

5. *Drainage.*—The drainage of the tunnel was effected by gravity, except for about 500 m. in the center of the tunnel.

6. *Transportation.*—The transportation-service was one of the most important portions of the tunnel-work. The motive-power used consisted at both ends of steam-locomotives, compressed-air locomotives, and horses. The steam-locomotives worked up to about 1,500 ft., and from this point the air-locomotives worked to within about 1,000 ft. of the headings; the remaining haul being done with horses.

7. *Method of Construction.*—The distance between portals is 12.4 miles, and except for a short curve at each end, the alignment is straight. The elevation of the Swiss portal is 2,250 ft. and of the Italian portal 2,076 ft. above sea-level; the highest point in the tunnel is midway between the portals, and is at an elevation of 2,310 ft.; from this summit-level the line descends on a 2-per cent. grade to Brigue, and on a 7-per cent. grade to Iselle on the Italian end.

The work consists of twin single-track tunnels exactly parallel in plan and profile, and lined throughout with masonry. The centers of the tunnels are 55.76 ft. apart; at the summit-level the cross-section is increased in dimensions to accommodate two tracks.

A center bottom drift was first driven by power-drills, and then timbered and covered with a closely-boarded roof. From this drift a shaft was driven upward to the roof-line every 164 ft. (50 m.). The top heading was then excavated by working in both directions from each of these shafts. Next in order, the floor of the upper heading was removed and then the two side cheeks of the bottom drift. The lower drift being timbered, no interruption of the traffic in it was caused by the removal of the rock above.

8. *Drilling.*—The advance drift was the only part of the operation performed by power-drills. The drills employed were Brandt rotary machines mounted in groups of two on a heavy thrust-bar about 12 in. in diameter. This thrust-bar was pivoted to a drill-carriage and was counter-balanced.

The section at the heading was nominally 6.5 by 9.5 ft., or 61.75 sq. ft., and as the depth of each blast was roughly 4.5 ft., the material removed by each blast ranged from 265 to 275 cubic feet.

The average daily advance was about 16 ft. at the Italian end and from 20 to 21 ft. at the Swiss end. This work was in gneiss rock. In rock of more friable nature, such as anhydrite or calcium sulphate, an advance of as much as 34 ft. in 24 hr. was made. After each blast, the time required to clean the heading, set the drills, complete the boring, and remove the drill-carriage, was more than an hour.

9. *Explosives.*—The explosives used were dynamite at the Italian end, and blasting-gelatine at the Swiss end.

The dynamite was put up in packages of about 1 lb. in weight; each hole was charged with six cartridges; each blast in the drift, therefore, used from 60 to 66 lb. of powder, or about 6.5 lb. to the cubic yard. Charges were fired by ordinary fuses, cut so as to give an interval between the firing of successive holes; about 15 min. was required after each blast to clear the fumes from the heading. This was accomplished by means of a ventilating-pipe running close to the face and the use of a spray of water. The ventilating-pipe exhausted about 35 cu. ft. of air per sec., and the spray absorbed the gases.

10. *Mucking.*—The spoil was cleared from the face by one gang while another gang loaded the collected muck into narrow-gauge cars hauled by horses. No machines were used, all the material being handled by manual labor. The work of clearing the heading was rushed to enable the drills to be put to work as soon as possible. To this end the clearing-gangs were composed of men who had been previously rested by performing light work only, and only the most skilled and energetic laborers were employed. The majority of the workers were from southern Italy. There were 14 or 15 men at each heading, worked in three shifts daily. Each gang had two horses

for each shift. Horses, which cost $1.60 per 8-hr. shift, died off rapidly and were paid for by the tunnel-contractors. Other methods of transportation were tried but proved less economical than the use of horses in the advance headings. The horses took the cars to the compressed-air locomotives, and these in turn took them to the steam-locomotives, as already described.

11. *Geological Conditions.*—The materials penetrated, beginning at the working entrance at the Italian end, are as follows: At the entrance a crust of quartz; following this for 14,268 ft. was a very hard gneiss lying in horizontal strata and known as antigoria; the gneiss contained occasional seams of crystalline rock, quartz, sulphur, pyrites, etc. Beyond for a distance of about 130 ft. a disintegrated slate clay was encountered, which proved a most treacherous material and was the most difficult part of the tunnel. Succeeding the disintegrated slate for about 200 ft. was a mixture of mica-schist, schistic gneiss, cipolin, quartz, and white marble, followed by about 325 ft. of anhydrite or crystalline calcium sulphate; then followed calcareous rock, schists, anhydrite, granitoid rock, and schistic gneiss, the last being nearly as dense and hard as the antigoria first encountered. All these formations were in horizontal strata.

To pass the disintegrated clay and slate a method of steel-and-timber strutting was employed.

In antigoric gneiss the time taken for each portion of the attack was as follows:

Bringing up and adjusting drills,	20 min.
Drilling,	1 hr. 45 min., to 2 hr. 30 min
Charging and firing,	15 min.
Clearing away débris,	2 hr.

or for one whole attack, between 4.5 and 5.5 hr., resulting in an advance of 3 ft. 9 in., or a daily advance of 18 feet.

From this it appears that the time spent in clearing away the spoil equaled that consumed in drilling, and it is in this clearing that a saving of time is likely to be effected rather than in the process of drilling.

The average temperature at the face was 73° F. during drilling-operations, 76° F. after firing, and a maximum of 80° F. on the south side, with 80° F. and 85° F. before and after firing.

12. *Rate of Progress.*—The progress for three months is given in the trial-report for 1900 as follows:

At Brigue, where there were three drilling-machines in one heading and two in the parallel heading, the total length excavated was 995 yd., or 6,409 cu. yd., in 89 working days The average cross-sectional area was 57 sq ft. This required 507 attacks and 3,066 holes, which had a total depth of 26,600 ft., and 14,700 re-sharpenings of the drilling-tool.

At Brigue 648 men and 29 horses were employed at one time in the tunnel. At Iselle the numbers were 496 men and 16 horses, working in shifts of 8 hr. Outside the tunnel in the shops, forges, etc, the men work 8 to 11 hr. per day, the total being 541 men at Brigue and 346 men at Iselle.

On the Italian side, where the rock is very much harder, there were three drilling-machines in each heading, the total length excavated, with a cross-sectional area of 62 sq ft., was 960 yd., or 6,700 cu. yd., in 91 working days. This required 61,293 re-sharpened tools, 758 attacks, 7,940 holes, with a total depth of 33,000 ft., and 56,000 lb of dynamite. The average time spent in drilling was 2 hr. and 55 min., and in charging and clearing, 2 hr. 36 min.

Thus in the hard gneiss, to excavate 1 cu. yd. of rock required 8.5 lb. of dynamite and each tool pierced 6.5 in of rock before it required resharpening.

IV. The Loetschberg Tunnel.

The Loetschberg tunnel, driven through the Bernese Alps, in Switzerland, is the last link of a railroad system connecting the city of Berne directly with the village of Brigue, which is situated at the north portal of the Simplon tunnel. With its completion, and the lately finished 12,000-ft. Weisenstein railroad-tunnel located about 30 miles north of Berne, it forms the shortest route between London, Paris, Brussels, or Hamburg, and Genoa, via Berne, Thun, Brigue, and Milan. A sketch-map of the Loetschberg railroad is given in Fig. 1, and a view from Brigue of the lower part of the road, under construction, is presented in Fig. 2.

The question of connecting the Bernese Oberland with the Rhone valley had its origin as far back as the year 1866, and the present location of the tunnel was proposed in 1899. In that year two consulting engineers, at the request of the Bernese government, began a careful study of the location of the proposed road, and reported in favor of a single-track tunnel 44,500 ft. long, basing their estimate on an average cost of \$4.90 per cu. yd. for tunnel-excavation and \$6.55 per cu. yd. for masonry lining throughout the tunnel-length. The total cost of the tunnel was estimated at \$107 per linear foot.

Assuming an average progress of 4 ft. per day for hand-

drilling, or from 5 to 8 ft. for machine-drilling, and from 15 to 18 ft. for rapid driving, the time required for driving the tunnel was estimated at 5 years. The maximum rock-temperature expected to be encountered was 95° F.

Later on, however, the expected increase of the traffic through the Simplon tunnel brought out the question of accommodating two tracks in the proposed tunnel, and therefore it was decided to drive a double-track tunnel.

Estimates were prepared, and the cost of the new proposed tunnel was calculated to be:

Tunnel-excavation and lining, ...	$8,660,000
Tracks, installations, etc., ...	1,400,000
Total, ...	$10,060,000

or a total cost of $211 per linear foot.

The franchise for building the road was granted to the Canton Berne in December, 1899, and, aside from 6,000,000 francs furnished by that canton, the necessary capital for building the road was obtained from French bankers.

The main offices of the company, Die Berner Alpenbahn Gesellschaft, are situated in Berne, while the main office of the contracting company is in Paris. A. Zollinger is the chief engineer.

The chief engineer for the company on the north portal is Mr. Von Erlach, and for the contractor Mr. Rothpletz; while for the south side Mr. Imhof is chief engineer for the company and Mr. Moreau for the contractor.

The new road begins at Frutigen, in the Bernese Oberland, about 32.5 miles from the north portal; 50.5 per cent. of this length is on horizontal curves, and 90 per cent. is on grade. There are 12 tunnels, aggregating 16,000 ft., one of which is a spiral tunnel 5,460 ft. long, with a 985-ft. radius. The maximum grade is 2.7 per cent., and the difference in elevation between Frutigen and the north portal of the Loetschberg tunnel is 1,370 feet.

The main tunnel is 47,678 ft. long, and was first planned to be on a tangent. After the cave-in of July 24, 1908, it was found necessary to insert a curve of 3,600 ft. radius in the tunnel in order to drive through solid rock. The elevation of the north portal is 3,940 ft.; of the south portal is 4,000 ft., and the

maximum elevation of the tunnel is 4,081 ft. above sea-level. The maximum grade in the tunnel is 7 per cent. The distance from the south portal to the end of the line, at Brigue, is 15.75 miles. Of this length 28 per cent., or 23,200 ft., consists of 21 tunnels, the longest being 4,450 ft. Of this stretch 54 per cent. is on curves, and 90 per cent. on grade. The difference in elevation between the south portal and Brigue is 1,110 ft., and the maximum grade is 2.7 per cent. The general profile from Spiez to Brigue is shown in Fig. 3.

Summarizing, the total length of the road is 45.8 miles, of which 36 per cent., or 86,900 ft., is tunnels. Let it be added that, for construction purposes, a narrow-gauge railway had first to be built to reach both portals. On the south side

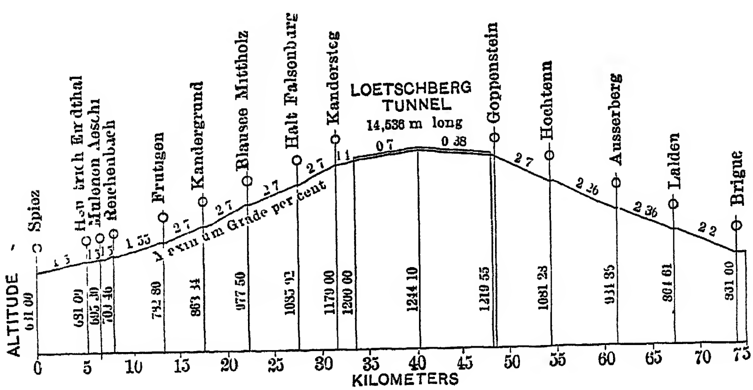


FIG. 3.—GENERAL LONGITUDINAL PROFILE FROM SPIEZ TO BRIGUE. SCALE, 1 : 800,000 HORIZONTALLY ; 1 : 40,000 VERTICALLY.

of the tunnel the construction-railway necessitated 38 tunnels, aggregating 18,000 ft. Of the 38 tunnels, 11 only will be part of the permanent road. A view of the construction-plant at Naters at the foot of the Loetschberg railroad, giving a glimpse of Brigue and the Simplon railroad, is given in Fig. 4, and that at Goppenstein, at the south portal of the tunnel, in Fig. 5.

1. *Geological Conditions.*—Beginning at the north portal, the materials penetrated have been calcareous for a distance of about 13,100 ft.; while on the south side, crystalline schist has been found on about 13,000 ft., and granite, forming the central part of the mountain, has been penetrated by both headings. Fig. 6 is an idealized section near the tunnel-axis.

But little water was encountered in the south heading, about 20 gal. per sec.; while in the north heading, 105 gal. per sec. necessitated the construction of quite a large-sized drainage-ditch.

On July 24, 1908, when the main heading had reached a point 1.6 miles from the portal, it struck a cleft filled with sand, gravel, and water. There was a sudden and violent inburst of these materials, which in a few moments filled up the tunnel for a length of 5,900 ft., burying 25 workmen and all the drills and other installations beyond hope of recovery. It is estimated that about 8,000 cu. yd. of sand and gravel entered the tunnel.

To avoid any further irruption of the materials, the tunnel was walled up by a 33-ft. wall at a point 4,675 ft. from the portal.

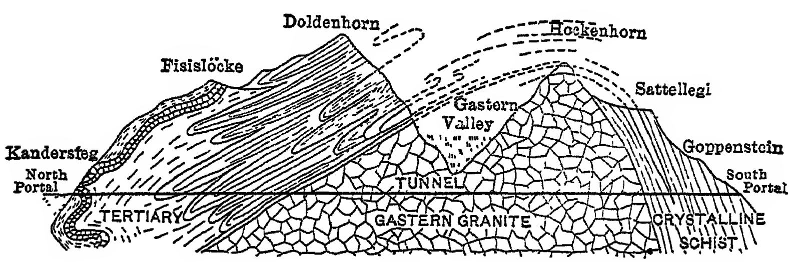


FIG. 6.—IDEALIZED GEOLOGICAL CROSS-SECTION NEAR THE TUNNEL-AXIS, LOETSCHBERG TUNNEL.

A commission of engineers was convened to decide upon a course to be adopted. Three methods were considered: (1) To force the tunnel through on the original line; this was considered impracticable, due to the great pressure from the 590 ft. depth of water, sand, and gravel over the tunnel. (2) To use the freezing-process; this also was considered impracticable. (3) To deviate the line and cross the Gastern valley further up stream. The last plan was adopted, and is shown in Fig. 1. The new line leaves the original location at a point 0.75 mile from the north portal. No further serious difficulty was experienced in tunneling through the diversion.

2. *Rock-Temperature.*—The usual high rock-temperature met during the construction of the Simplon tunnel was a serious hindrance to rapid driving. Careful studies were therefore

made in order to determine the maximum temperature to be expected in driving the Loetschberg tunnel.

As above stated, preliminary studies had fixed this maximum rock-temperature at 95° F. It was expected that this temperature might be slightly exceeded, and due provision was made for taking care of it. With 86 per cent. of tunnel completed the maximum rock-temperature recorded has been 91° F., and it is not expected that this figure will be very much exceeded.

The following rock-temperatures have been observed.

North Side.		South Side	
Kilometers.	Degrees Fahr	Kilometers.	Degrees Fahr
2	58	1.5	68
2.5	58.5	2	75
3	61	2.5	77
3.5	56	3	79
4.5	60	4	82
5	61	4.5	87
5.5	66	5	90
6	69	5.5	91
		6	93.5

3. *Rate of Progress.*—Driving of the headings was begun on Oct. 1, 1906, for a single-track tunnel, and continued until Oct. 1, 1907, when it was decided to drive a double-track tunnel; 86 per cent. of the tunnel had been driven by Oct. 31, 1910. The headings met Mar. 31, 1911. On Oct. 31, 1910, the 4,000 ft. of heading which had been abandoned after the cave-in of 1908 had been regained.

4. *Power.*—The power-plant for the south heading is situated at Goppenstein. It is driven by electric power. The current is brought at 15,000 volts, and stepped down to 500 volts for power-purposes.

Compressed air for the drills (Ingersoll-Rand) is furnished by 3 two-stage Ingersoll-Rand compressors, each having a capacity of 1,950 cu. ft. of free air per min., and a compression of 145 lb. per sq. in. They are driven by 400-h-p. electric motors. Compressed air for the locomotives is furnished by 2 four-stage Ingersoll-Rand compressors, having a capacity of 460 cu. ft. of free air per min., and a compression of 1,760 lb. per sq. in. They are driven by 250-h-p. electric motors.

The power-plant for the north heading is situated in Kandersteg. Electric power, used throughout the works, is brought

FIG. 1.—MAP SHOWING THE LOCATION OF THE LOETSCHBERG RAILROAD. THE LIGHT DOTTED LINES INDICATE THE FIRST PROPOSED TUNNEL-AXIS, BEFORE THE CAVE-IN.

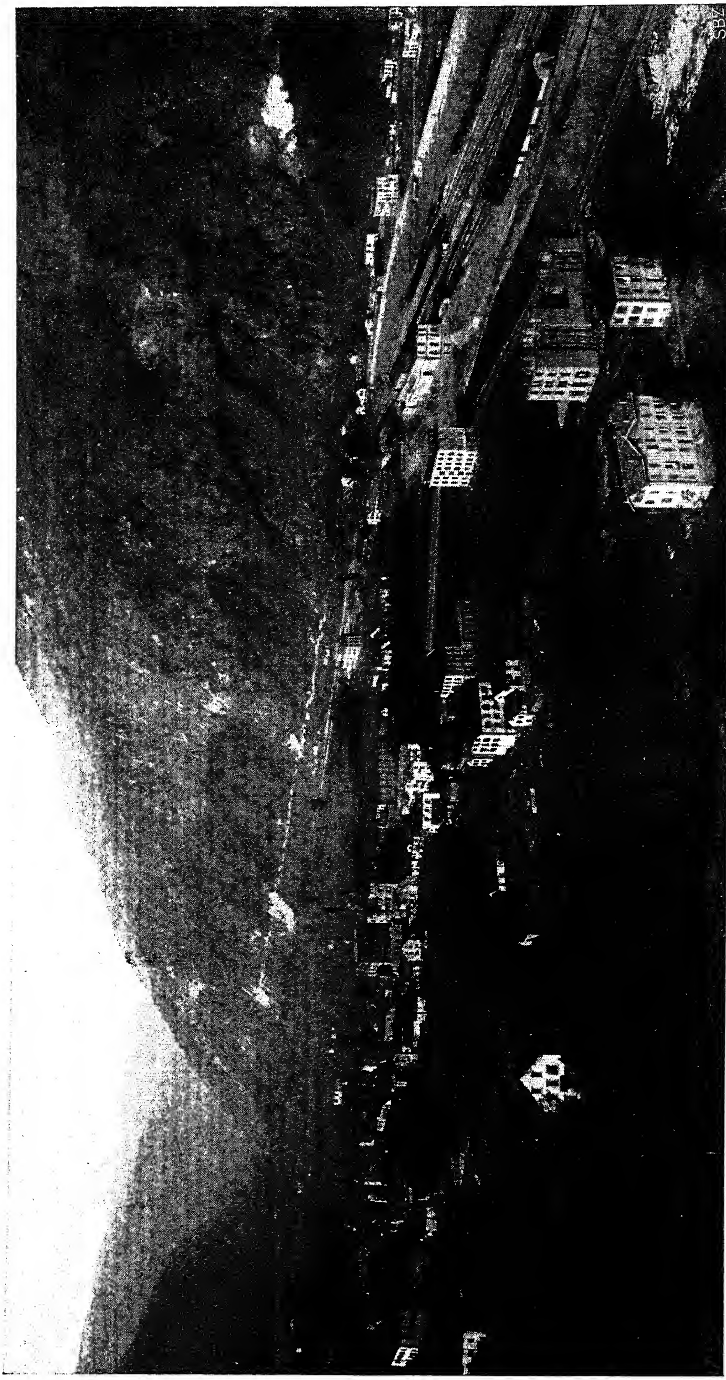


Fig. 2.—View from Brigue of the Lower Part of the Loetschberg Railroad (Under Construction).

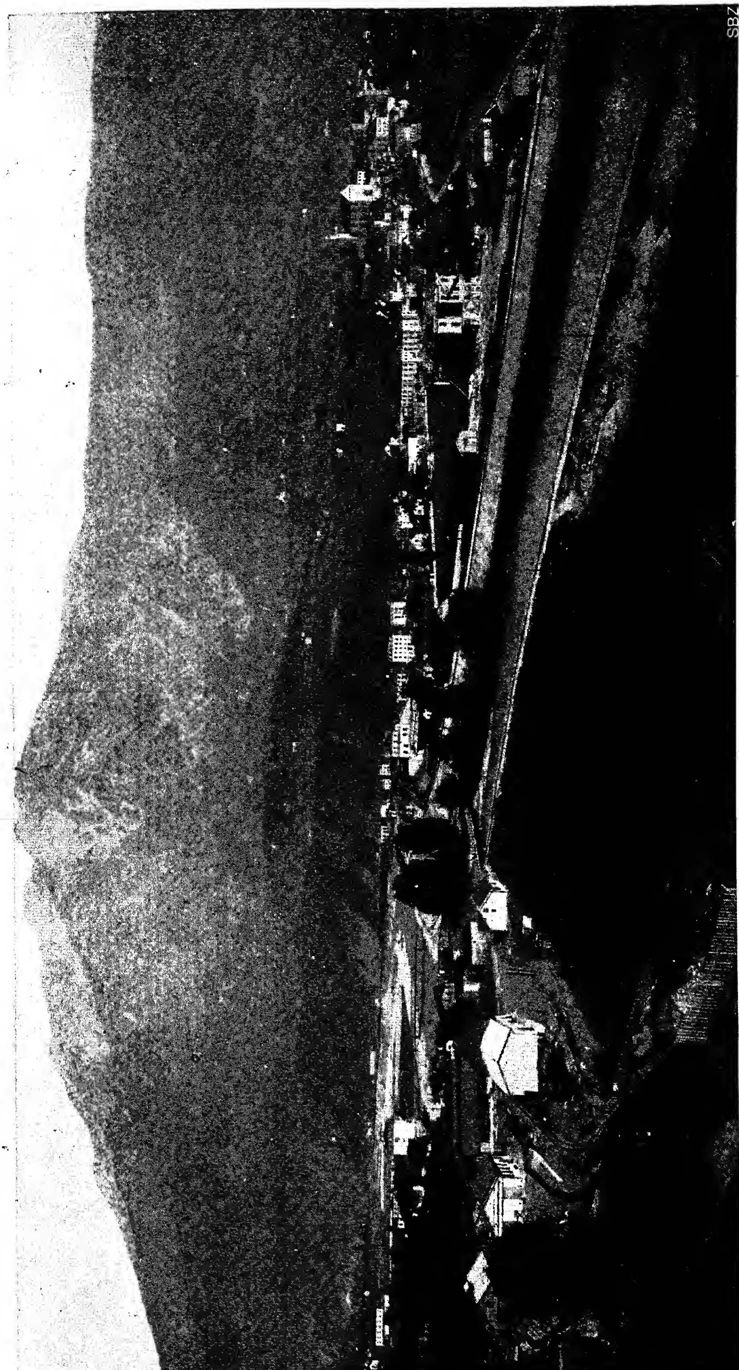


Fig. 4.—View of the Construction-Plant in Naters, at the Foot of the Ascent of the Loetschberg Railroad, with Glimpse of Brigue and the Simplon Railroad.

FIG. 5.—CONSTRUCTION-PLANT AT GOPPENSTEIN AT THE SOUTH PORTAL OF THE LOETSCHBERG TUNNEL.

FIG. 7.—COMPRESSOR-ROOM IN KANDERSTEG.

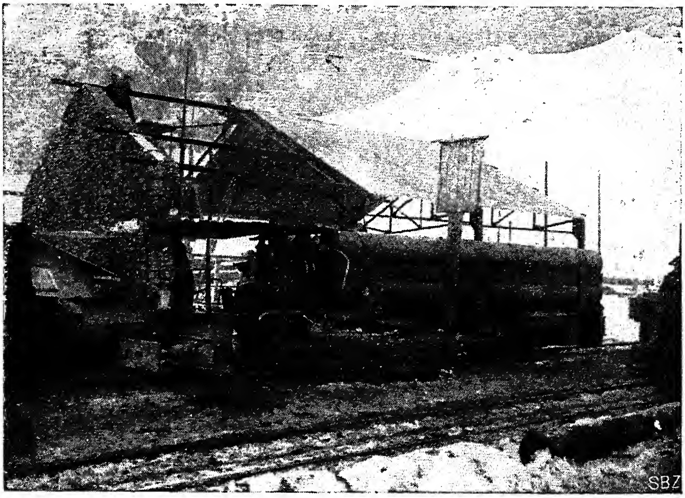


FIG. 8.—COMPRESSED-AIR TUNNEL-LOCOMOTIVE.

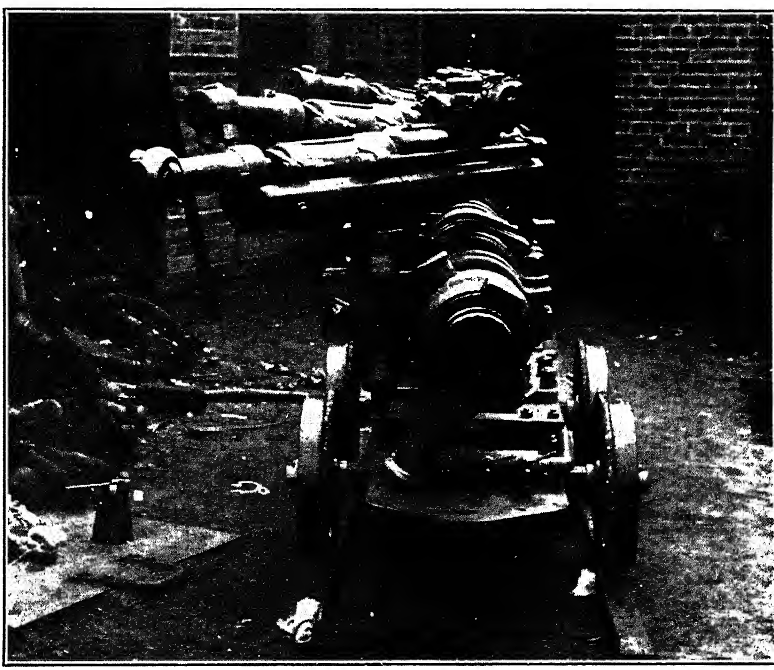


FIG. 10.—TUNNEL-CAR WITH DRILLS MOUNTED IN POSITION TO BE TAKEN INTO THE HEADING AFTER A BLAST.

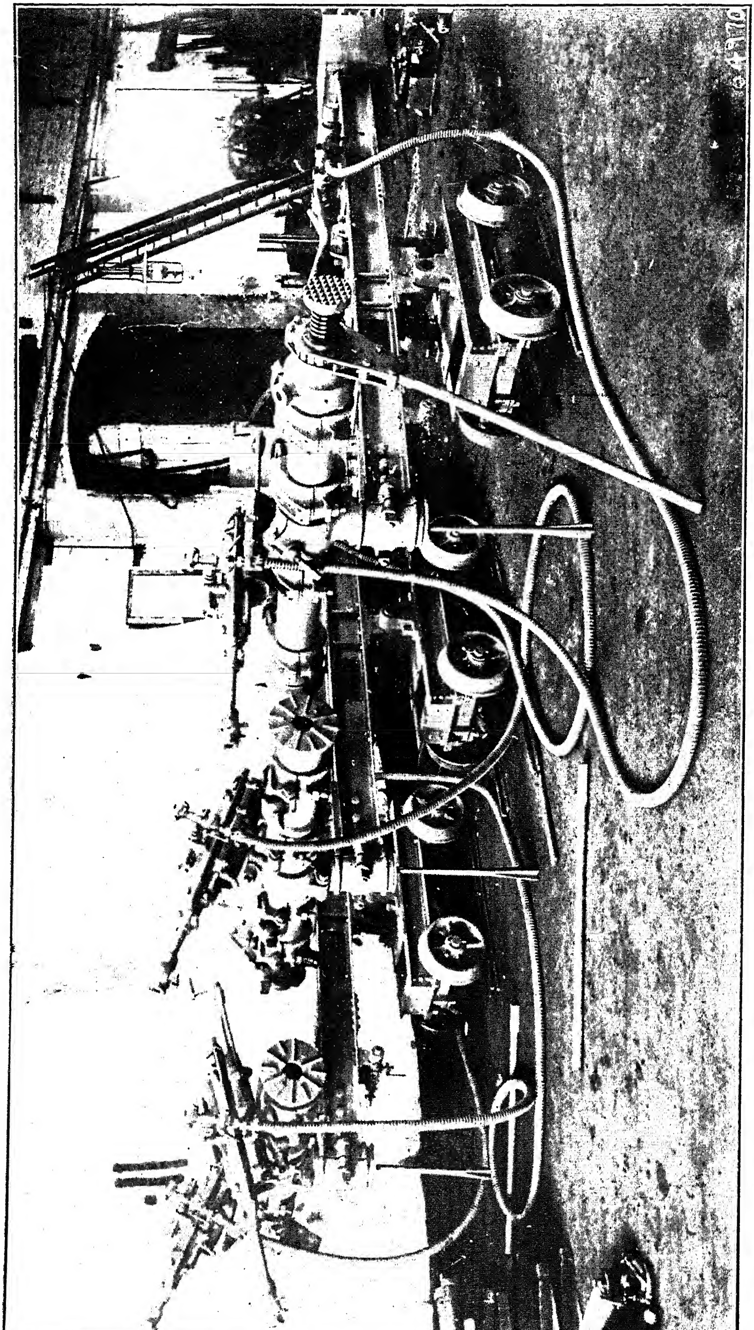


FIG. 9.—DRILL-CARRIAGES FIRST USED AT LOETSCHBERG, THE CAR CARRYING A BEAM WITH A COUNTERWEIGHT.

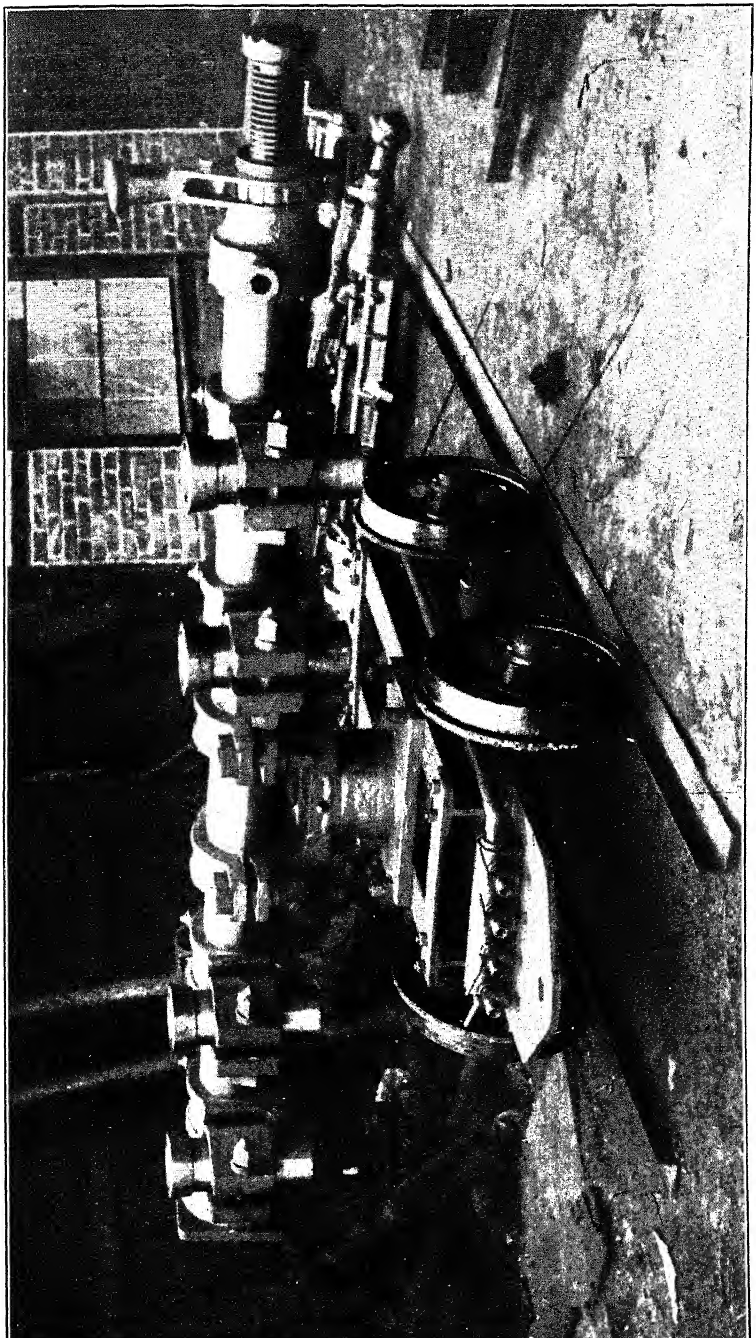


Fig. 11.—Tunnel-Car for Carrying Four Drills; now in Use in the Loetschberg Tunnel.

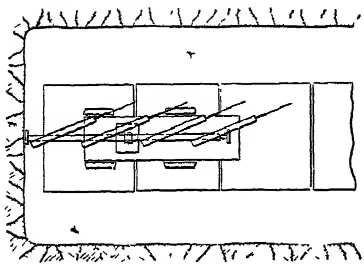


FIG. 12.—CARRIAGE BROUGHT FORWARD AFTER MUCKING.

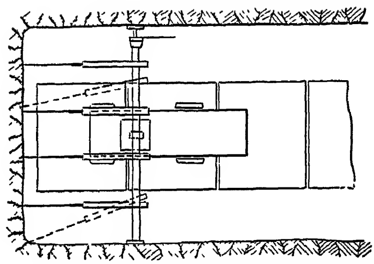


FIG. 14.—HORIZONTAL ADJUSTMENT OF DRILLS.

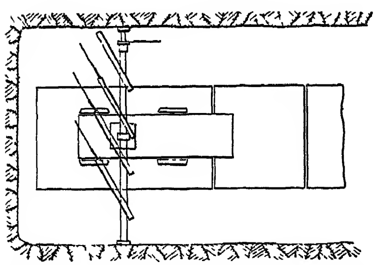


FIG. 13.—DRILL-SHAFT IN POSITION TO BE JACKED.

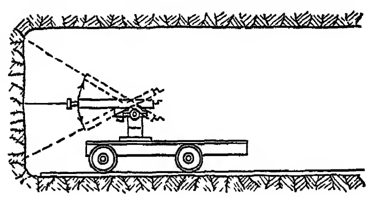
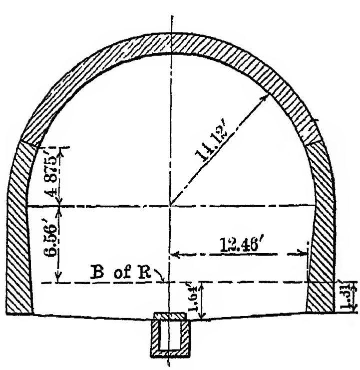
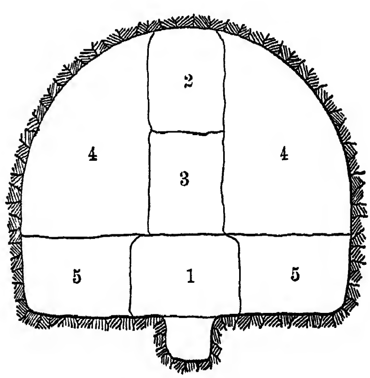


FIG. 15.—VERTICAL RANGE OF DRILLS.



Standard Section.



Excavation-Diagram.

FIG. 16.—STANDARD SECTION AND EXCAVATION-DIAGRAM OF LOETSCHBERG TUNNEL.

from Spiez at 15,000 volts, and stepped down to 500 volts for power-purposes in the tunnel as well as in the shops.

Compressed air for the drills (Meyer) is furnished by two units, each consisting of a two-stage Meyer air-compressor, each having a capacity of 1,770 cu. ft. of free air per min., and a pressure of 117 lb. per sq. in. They are belt-driven by 450-h-p. electric motors. A view of the compressor-room at Kandersteg is given in Fig. 7.

Compressed air for the locomotives is furnished by two units, each consisting of a five-stage Meyer high-pressure compressor, with a capacity of 565 cu. ft. of free air per min., and a pressure of 1,760 lb. per sq. in. They are belt-driven by a 250-h-p. electric motor. Fig. 8 is a view of a compressed-air locomotive.

5. *Transportation.*—All trains in the tunnel are operated on a regular time-schedule, changed from time to time according to the progress of driving.

Steam-locomotives are used outside, while compressed-air locomotives are run inside of the tunnel. A few mules are still in use in the south bottom-heading.

Four types of cars are used for the service inside and outside of the tunnel: (1) Passenger-cars having a capacity of 24 men each, and run only when shifts are leaving or entering the tunnel. (2) Cars having a capacity of 35 cu. ft., used for mucking. (3) Cars of 70 cu. ft. capacity, used chiefly for transporting masonry. (4) Flat cars, used for bringing in timber, rails, etc. The cars used for mucking are 6 ft long, 2 ft. 8 in. wide, and 2 ft. deep, the upper edge being 3 ft. 7 in. above the top of the rail.

The gauge for all tracks laid in the tunnel is 30 in.; the rails, of from 30 to 40 lb. per yd., being laid on wooden stringers, except in the last 100 ft. of the bottom heading, where portable rails with pressed-steel ties are used. Trains are run in the tunnel at a speed of from 8 to 10 miles per hour.

6. *Lighting.*—Electric light is used only in that part of the tunnel already lined with masonry and partly completed.

Portable acetylene-lamps are used throughout the tunnel with a few exceptions only. They are sold to the men by the contractors for the sum of $1. Besides giving a very bright light, these lamps are clean, easily handled, and they do not give out fumes as do oil-lamps.

7. *Drainage.*—Drainage on the north side of the tunnel is provided by means of a ditch 2 ft. 8 in. wide, and 2 ft. deep, placed between the two tracks. It has the same slope as the tunnel. The flow of all springs encountered on this side of the tunnel amounts to about 105 gal. per sec., most of the water coming from that part of the tunnel where the cave-in occurred in 1908. The flow on the south side of the tunnel amounting to only 20 gal. per sec., the section of the drainage-ditch is but half the size of the one above named.

8. *The Drill-Carriage.*—The records made in driving the headings are due to the excellent organization, and to the methods of setting up and taking down the drills.

Fig. 9 shows the type of drill-carriage first used at Loetschberg, carrying a beam with a counterweight. Fig. 10 is a view of a carriage with drills mounted in position to be taken into the heading after a blast, and Fig. 11, the carriage carrying four drills now in use at the Loetschberg tunnel. In this type the beam and counterweight are omitted, the bar, on which four, five, or six drills are mounted, being placed directly on the truck. The width of the tunnel in which these cars can operate varies from 6 to 13 feet.

A drill-carriage of simple but efficient design was devised by the contractors. Each carriage carries four or five drills. Fig. 12 shows the carriage, together with the drilling-machines, when brought forward just after mucking in the heading. Fig. 13 shows the horizontal shaft swung into position ready for being jacked, and the drills ready to be swung into the position shown in Fig. 14. It can be easily seen from Fig. 14 that the drills can be independently swung through an arc of a circle or moved sideways, while in Fig. 15 the different positions which the drills can be given by being swung in a vertical plane are shown.

The time required to change the machine from the position shown in Fig. 12 to that shown in Fig. 14 and to commence drilling is usually from 6 to 8 min. This fact alone shows the superiority of this system of carrying the drills for such work over any other method used up to the present time.

9. *Explosives.*—Three kinds of explosives are used. Dynamite, with about 85 per cent. of nitro-glycerine, is mostly used in

the headings. Westphalite and cheddite, being more safely handled, are used for enlarging and for small blasts.

Great stress is laid on the fact that a high-grade explosive breaks the rock into small pieces, not larger than an orange, which enables mucking to be done with shovels.

Firing is done with ordinary fuses. The dynamite cartridges are wrapped with red paper in order to be easily detected in case of a mis-fire. Dynamite-carriers and handlers are provided with red lanterns.

10. *Labor and Wages.*—Italian labor is used throughout the works with the exception of some Macedonians lately imported. Mostly Italians from the northern provinces of Italy are employed.

A bonus system of payment is used throughout the different kinds of operations. The following wages are paid:

	Daily Wages.	Average Bonus.	Total
Drill-foreman, ...	$1.50	$1.10	$2 60
Drill-runners, ...	1.00	0.70	1.70
Muckers, ...	0.80	0.50	1.30
Nippers, ...	0.70	0.30	1 00
Tracklayers, ...	0.80	0.15	0.95
Masons, ...	1.00	0.40	1.40

There are three 8-hr. shifts per day.

11. *Excavation.*—As shown in Fig. 16, the width of the finished tunnel-section is 28 ft. at the arch-springing and 25 ft. at the base of the rail. The arch is semi-circular, the crown being 20.7 ft. above the base of rail.

The sequence of excavation is illustrated by Fig. 17. A bottom heading 6.5 by 10 ft. is first driven several hundred feet in advance of the enlargement. Upraises are then driven from 500 to 600 ft. apart, and a top heading started back and forth. The top heading is then enlarged as shown by the sections in Fig. 17.

When the inclination of the strata is vertical or the formation is of a treacherous nature, the method illustrated by Sections B–B and E–E in Fig. 17 is used.

In the bottom heading the mining-operations proceed as follows: The drill-carriage is run forward from its siding close to the face of the heading, passing over 5 ft. by 5 ft. by $\frac{3}{8}$-in. steel plates laid on the floor of the heading for a length

of about 30 ft. Each plate is provided with 1-in. holes at the corners for ease in handling with a pick.

The water- and air-pipes laid on one side of the heading to about 40 ft. from its face are connected with the drill-carriage, and the drilling begins with the top holes. Water-sprinkling is frequently done, especially in starting the holes, in order to lay the dust.

Without interfering with drilling, mucking is going on just behind the drill-carriage, and the loaded muck-cars are run back to a siding, where trains of from 20 to 30 cars are formed and hauled out by air-locomotives.

Drilling being completed in the heading, the drill-carriage is run back to its siding, and the steel plates laid on the floor are covered with a layer of muck about 4 in. thick to prevent deterioration.

The bore-holes are then loaded and carefully tamped, and the last man to leave the heading, after firing the fuses, opens the air-pipe valve, the escaping air thus creating a cushion of fresh air from the face of the heading back to a certain distance, so that, after blasting, the muckers are able to go to work without delay.

A high-grade explosive only is used in the heading, which breaks the rock in small pieces and renders mucking with shovels easy. The bore-holes, having an average depth of about 4 feet, are started with a 3-in. drill and finished with a 2-in. drill. On account of giving better results, firing is done with fuses, about 4 ft. long, the center holes being fired first.

Mucking-operations proceed as follows: Two empty cars are run to the heading, the first one being immediately loaded by two or three men shoveling without interruption until the car is fully loaded. This operation is performed in 3 or 4 min., which means that 1 cu. yd. is loaded in from 2.5 to 3 minutes.

Owing to the manner of drilling and blasting and to the shallow holes, the muck, instead of piling up in front of the face of the heading, is thrown back, and forms a layer over the floor, which enables the track to be cleared rapidly.

Getting rid of the muck is always a problem in tunnel-driving. At Loetschberg a cubic-meter car (35.5 cu. ft.) is filled in 5 min., and it takes only 1 min. to get this car away and bring an empty car to the heading. In order to do this,

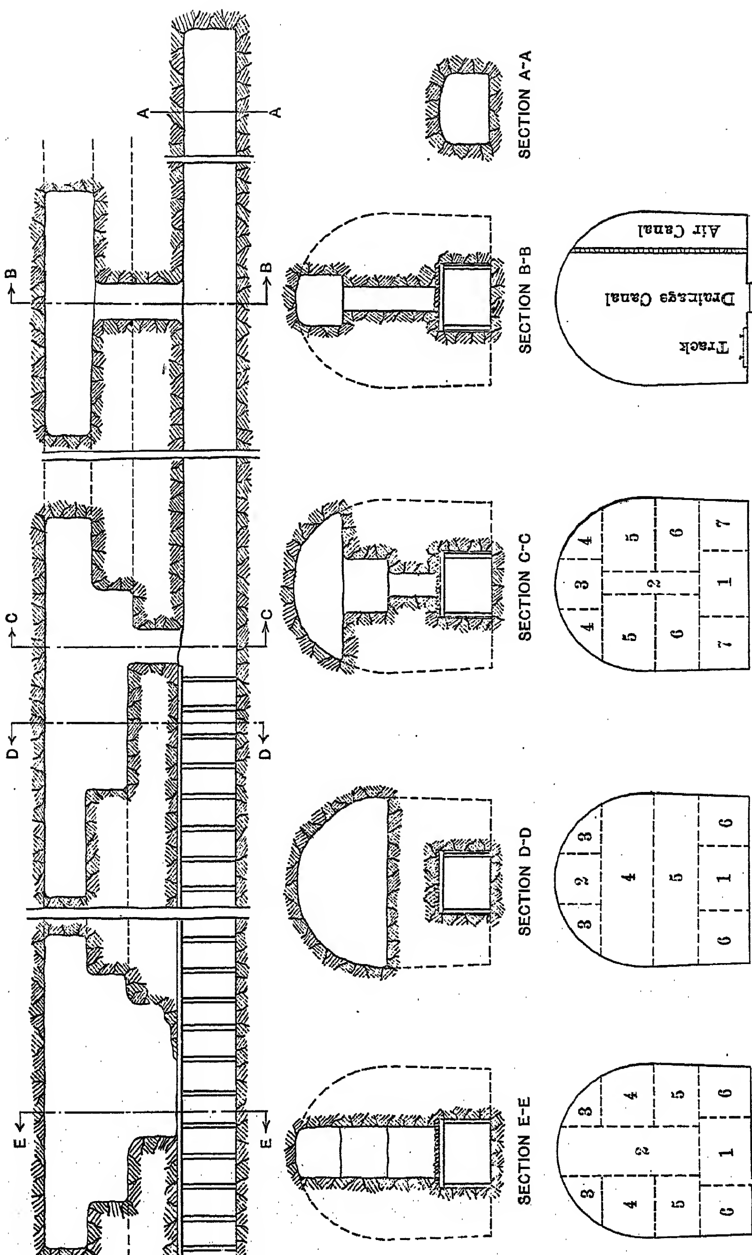


FIG. 17.—LONGITUDINAL AND CROSS-SECTIONS, SHOWING METHODS OF EXCAVATING.

small entries or chambers are excavated at intervals in the lateral wall of the main heading, which enable an empty car to be thrown from the track on the side, thus clearing the track and allowing the filled car to pass, whereupon the empty car is turned up on its wheels and rolled into the heading. Here we have an illustration of an improvised siding in a narrow heading, by means of which one car may pass another. This system is shown in Fig. 18, the operation being as follows:

When the car, *A*, is filled, it is taken away on the track, *B*, and immediately after it has passed the point, *C*, the empty car, *D*, which had been reversed on its lateral side, is thrown back on the track, brought to the advancement and filled again

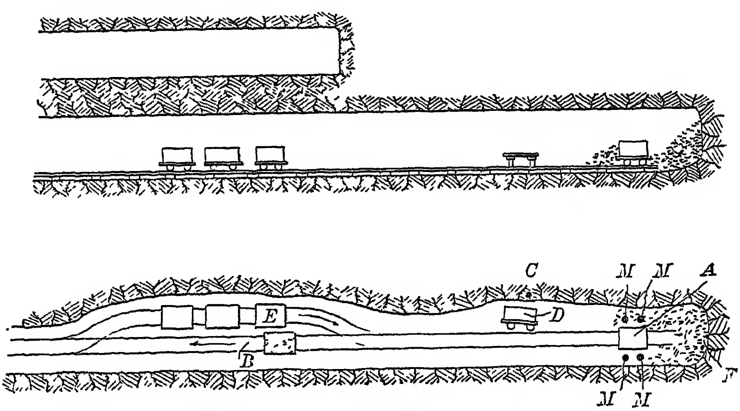


FIG. 18.—IMPROVISED SIDING IN NARROW HEADING.

in the space of one minute. As soon as car, *D*, has been brought to the advancement, another empty car, *E*, is brought to the same point, *C*, reversed on its side, and waits until car, *D*, is filled and taken away again, etc. In one instance, 14 car-loads, each of 1 cu. m. volume, were taken away in 1.5 hr., which cleared the heading completely and allowed the drill-wagon to be brought in. Ten men are busy removing the débris, two of which number get at the extreme limit marked *F*, and their work consists in searching the débris for the dynamite cartridges which might not have exploded. Of the remaining eight men, four work to fill the car, as shown at *M*, which takes 5 min.; they then rest for 5 min., while the second gang of four men come and fill the second car, etc.

Drilling is started not more than 5 min. after the removal of the last car-load. This result, which at first sight seems impossible, is only obtained by absolute discipline.

The man who knows that his only work at this moment is to connect the air-main to the drill-carriage does not do anything else; the men whose duty it is to screw the carriage tightly to the wall immediately jump to the right place.

The system has been adopted of low and wide gallery in the proportion of 1 : 2; the gallery being 6 ft. high by 12 ft. wide.

The rate of drilling is, 15 or 16 holes in 1.1 to 1.15 hours.

An engineer who recently visited this work says:

> "When I arrived at the heading it was 9.30 a. m. The holes were being prepared for blasting. The blast took place at 9.35 a. m.; 5 min. after the blast the men were in place removing the débris, and at a little after 11 a. m. the drill-carriage was in place again and the rock-drills were working. It usually takes from 25 to 30 min. between the time at which the drilling is finished and the time at which the start is made to remove the débris; that is to say, 25 min. for taking away the drill-carriage, cleaning the holes, loading with explosives and blasting. An additional 5 min. are consumed in getting the smoke away by means of the ventilator and then the men get to work at the débris. In order to assist the men a spray of water is discharged near the heading after the blast. This water is brought into the tunnel in a pipe placed within a larger pipe, which insulates it and keeps its temperature from being affected by the temperature of the tunnel."

Drilling in the top heading is accomplished by means of two or three drills carried on tripods or on a horizontal bar, while hammer hand-drills are used generally for the enlargement.

Mucking-operations in the top heading are very simple, since all blasted material is dumped directly through the upraises into cars running on a siding in the bottom heading.

The operations of blasting, mucking, timbering, and hauling are performed without interruption and without interference with each other, and a special force of engineers is required in order to obtain such a result.

All employees and workmen are insured against accident or death, by the contracting company, and great care is therefore exercised in handling explosives and in operating the trains. Data pertaining to driving the headings are given in Table I.

12. *Ventilation.*—Ventilation in the tunnel is obtained from two ventilators 11.5 ft. in diameter, having a capacity of 53,000 cu. ft. of air per min. at 5.5 oz. pressure. Each ventilator is belt-driven by a 175-h-p. electric motor, housed in a building

at each portal, forming part of the permanent ventilation-system.

Air is taken in that part of the tunnel already completed through a canal of 68 sq. ft. area, made of hollow tiles, shown in Fig. 17, then through steel pipes to electric-driven ventilators running in series, and having a capacity of 6,300 cu. ft. of air per min. at 14 oz. pressure. These two ventilators are mounted on carriages, and are moved along as the work advances.

Openings are provided at intervals in the above-described air-canal so as to allow part of the air to escape in the tunnel.

13. *Rock, Temperature, etc.*—In comparing conditions on the north and south ends of the Loetschberg tunnel, it is well to bear in mind that at the south end the rock is generally harder and the temperature higher. At the north end the temperature varies from 75° to 80° F., while at the south end it has reached a maximum of 110° F. It is also claimed that a remarkable organization of the working-force exists on the north end. The settlement at the south end was built by the Loetschberg Co. The winter there is extremely severe and dangerous. Three years ago, by an avalanche, seven men were killed, among them an American engineer named Merwarth, who was at the time installing the compressed-air machinery. Conditions of this kind do not favor the contractor in getting the best kind of labor. Kandersteg, at the north end, is practically a summer-and-winter resort, full of good hotels, easy to reach, and a more favorable labor-market.

At one time, when the north heading was making a daily progress in excess of the south heading, the contractors sent some of the drills from the north heading over to the south, thinking that perhaps the difference in progress was due to the drill, but the results were not changed. It seems plain that were it a question of machinery only, the machinery making the greater progress would be used throughout, but the difference appears to be one of natural conditions and of organization.

TABLE I.—*Loetschberg Tunnel—Drilling- and Working-Conditions.*

N, North Side (KANDERSTEG), S, South Side (GOPPENSTEIN).

(From Official Report of the Berner Alpine Railway.)

QUARTERLY REPORT, No.	6		7		8		9		1908.	
	Jan.-Mar.		Apr.-June		July 1-July 24 Sept. 30		Oct.-Dec.		Yearly Average.	
	N.	S.	N.	S.	N.	S.	N.	S.	N.	S
	Mar. 8–10 —Work Stopped, Rock-Conditions Extremely Favorable. Mountain Limestone	Feb 20–Mar. 31 —Work Stopped, Avalanche Accident	Mountain Limestone	Ventilation Only One-half that on North Side, Chlorite and Sericite Gneiss.	July 24 —Cave in at Kander, 25 Men Buried		Tunnel-Driving Stopped	Gneiss.		
Progress of heading, feet	1,668	807	2,010	1,618	430	1,751	...	1,506	4,108	5,682
Average cross-section of heading, square feet	64.52	62.40	63 48	68.88	63.48	63.48		61.52	65 60	64 52
Cu yd broken out of heading	3,975	1,879	4,699	4,125	998	4,141		3,610	9,672	13,755
Number of working-days	...	...	83	89	84	90½	...	88½		
Average daily progress, feet	18.85	14 79	25 78	18 30	18 66	19 34		16.92	21 09	17 33
Average progress of one round, feet	3 83	3 91	4 19	3.54	3 83	3.44	..	3 18	3 95	3 51
Number of rounds	457	206	505	459	112	510	..	473	1,074	1,648
Drilling-time of one round, hours	1 40	2.20	1.17	1.35	1.23	1 15	..	1.29	1 27	1.40
Mucking-time of one round, hours	2 45	3 20	1.37	2.53	3.10	2 55	...	2 51	2.30	3.00
Entire duration of one round, hours	4.35	6 22	3.58	4 40	4 34	4 15		4 31	4 22	4 58
Number of bore-holes per round	12 5	11 4	12 8	12 0	12 5	11 7	..	12 0	12.6	11.8
Average length of holes, feet	4.13	4.82	4 46	4 62	4 43	3 54	..	4 33	4 33	4 33
Length of holes per cu. yd., feet	4.75	5 82	6 15	6 18	6 25	5 04	...	6 63	5.72	5 92
Dynamite per cu. yd, pounds	6.60	5.66	6.38	6.05	5.88	5.94	...	7.05	6.28	6 17
Number of steel bits per cu yd.	2 57	1.27	1.86	1.34	1.45	2 85	..	3.57	1.96	2.26
Number of drills in operation	3 2	3.2	3 4	4	3	4	.	4	3 2	3 8
Air-pressure at compressor, pounds	...	...	113	99 3	110 8	95	113	90.8	113	99 3
Air pressure at face, pounds	78	85.2	106 5	85 2	106 5	72 6	106.5	71	99.3	78 1
Air-temperature at face, degrees Fahrenheit	53	71 6	55 4	75 4	51.4	76.6	...	80.6	53.2	76
Rock-temperature at face, degrees Fahrenheit	...		...	.	..	...	...	78 8		

TABLE I. *(continued)—Loetschberg Tunnel—Drilling- and Working-Conditions.*

N, North Side (KANDERSTEG), S, South Side (GOPPENSTEIN).

QUARTERLY REPORT, No.	10		11		12		13		1909		14		15		1910.	
	Jan. 1–Mar. 31. Feb 22–Mar. 31.		Apr.–June.		July–Sept		Oct.–Dec		Yearly Average.		Jan.–Mar.		May–June		First Six Months.	
	N	S.	N.	S	N	S	N	S	N	S	N.	S	N	S	N	S
	Mountain Limestone.	Dolomite, Slate, Etc.	Black Mountain Limestone	Various Sedimentary Limestones, Slate and Granite.	Mountain Limestone.	Granite Laccolithen Gastern Granite	Malm Trias	Porphyry			Granite	Granite and Quartz-Porphyry	Gastern Granite.	Granite and Quartz-Porphyry	Gastern Granite.	Granite and Quartz-Porphyry
Progress of heading, feet	937	1,402	2,775	1,402	2,848	1,562	985	1,422	7,550	5,772	2,250	1,450	2,490	1,500	4,740	2,950
Average cross-section, square feet	70.15	69.90	71.00	66.7	67.0	67.2	83.9	66.7	73.0	67.6	70.8	66.5	67.6	68.5	69.2	64.6
Cu yd broken out	2,437	3,550	7,285	3,460	7,080	3,890	3,047	3,510	19,849	14,410	5,890	3,570	6,230	3,530	12,100	7,100
Working-days	36½	90	85½	87	84	90	53½	88½	259½	355½	86½	84½	86	90	172½	174½
Average daily progress, feet	25.7	15.5	32.3	16.2	33.9	17.3	18.5	16.0	a27.6	16.3	26.0	17.15	28.9	16.65	77.4	16.9
Average progress, one round, feet	1.23	3.18	4.82	3.41	4.85	3.70	4.33	4.16	4.55	3.62	4.13	3.97	4.07	4.2	4.1	1.07
Number of rounds	221	440	573	408	588	420	227	342	1,609	1,670	541	366	613	358	1,157	724
Drilling-time, one round, hours	1.16	1.35	1.05	2.18	1.03	2.22	1.14	2.54	1.12	2.17	1.18	2.31	1.13	2.48	1.15	2.40
Mucking-time, one round, hours	2.35	3.04	2.26	2.40	2.20	2.37	2.50	3.09	2.33	2.53	2.19	2.50	2.02	3.10	7.10	3.00
Entire duration, one round, hours	3.54	4.39	3.35	5.07	3.26	5.08	5.39	6.13	4.08	5.18	3.50	5.32	3.22	6.02	3.36	5.47
Bore-holes per round	14.5	12.7	14.24	13.72	14.49	12.77	14.1	14.0	14.3	13.3	14.1	14.0	14.89	14.3	14.5	14.15
Average length of holes, feet	4.72	4.16	5.01	4.26	5.05	4.36	4.72	4.52	4.87	4.32	4.76	4.43	4.56	4.69	4.66	4.56
Length of holes per cu. yd, feet	6.25	6.60	5.64	6.91	6.11	6.08	4.99	6.20	5.74	6.44	6.24	6.49	6.71	6.84	6.48	6.62
Dynamite per cu. yd, pounds	6.57	7.64	5.53	8.19	5.60	7.36	4.21	6.79	5.48	7.48	6.17	7.09	6.94	6.88	6.53	6.98
Steel bits per cu. yd	0.85	3.66	0.72	5.42	0.65	6.59	0.66	5.85	0.72	5.38	2.83	6.42	4.32	6.83	3.57	6.62
Drills in operation	3.98	4.0	4.0	5.1	4.0	4.9	4	4	4	4.5	4	4	4	4	4	4
Air-pressure at compressor, pounds	113.6	78	106.5	79.5	106.5	88	106.5	110.8	108.2	89.5	121	96	121	99	121	98
Air-pressure at face, pounds	106.5	63.9	99.4	71	97.9	65.3	99.4	63.5	100.8	65.9	106	71	114	92	110	81
Air-temperature at face, degrees Fahrenheit	50.3	80.5	60.4	81.5	63.2	82.4	59.5	82.7	58.3	81.8	59.4	83.7	66.2	86.6	62.8	85.2
Rock-temperature at face, degrees Fahrenheit	48.2	78.8	56.8	81.7	59.4	86.0	56.1	86.5	55.1	83.2	60.1	90.3	61.7	97.3	60.8	91.4

NOTE.—On the south side, after blasting, 12 connection-valves are opened for the rapid removal of the air in the tunnel During drilling ventilation is also done with compressed air

a This figure works out 29 1 when dividing total progress by number of days.

The official reports show that the quantity of air used for ventilation on the southern end was only about half that on the northern end, and it is generally said that the southern ventilating-system was not as effective as the northern, whereas the temperatures on the southern end were higher. The operating-force was compelled to use compressed air from the air-mains to increase the ventilation, which, in a measure, reduced the temperature, and decreased the pressure used for drilling. It is likely that in this way the efficiency of the drills may have been somewhat reduced. The working-pressures were: north end, 7.2; south end, 5.2 atmospheres.

The nature of the rock was different on the two ends of the tunnel, as shown in Table I. An average of one steel was required in the north end for 1 cu. m. of excavation, while on the south end an average of from 5 to 7 was required for the same work.

The average consumption of steels for 2.5 years was: north end, 2.33; south end, 7.70 steels per cu. m. driven.

The report for the year 1910 shows that in the first part the rock encountered on both the north and south ends was practically the same, although the average drilling-time was much less on the north than on the south end. To explain this, the air-pressure on the north end during this period was 7.75, as compared with 5.7 atmospheres on the south end, which largely accounts for the difference. It must also be noted that the number of steels per cubic meter of rock removed was, on the north end, 4.65; and on the south, 8.61, which indicates either that the effect of the rock on the drill-bits was different, or that the blacksmith-work was unequal.

I wish to express my indebtedness to Eugene Lauchli, the able Swiss engineer, now living in New York, who has greatly aided in the preparation of this paper. I am also indebted to G. H. Gilbert; F. A. Choffel, of Paris; and to Dr. Henry S. Drinker, through his work on *Tunneling;* to C. R. King, in *Engineering News; Tunneling*, by Prelini; *Practical Tunneling*, by Sims and Clark; *Modern Tunnel Practice*, by Stauffer; and to various encyclopedias and published articles relating to the Alps.

Mining-Costs at Park City, Utah.

BY FRED T. WILLIAMS.

(Wilkes-Barre Meeting, June, 1911.)

INTRODUCTION.

THE Park City mining-district is distinctively a camp of few properties, 5,000 acres, or one-third of the entire district, being under the management of but three companies. As a rule, the ore-bodies lie deep, with no outcrop, except where erosion has formed the deeper canyons and gulches. Ontario canyon exposed the famous Ontario ledge, and Woodside gulch showed the first ore of the Silver King Coalition mine. The Quincy ore-bodies were first brought to light in upper Empire canyon, and Thaynes canyon was instrumental in aiding the prospector during the early days of the camp. There are 22 shafts which have reached a depth of at least 500 ft., ten a depth of 1,000 ft., five a depth of 1,300 ft., and two a depth of 2,000 ft. The amount of lateral development has been proportionally extensive, including six long tunnels, four of which are in 3 miles, with a fifth now being driven. The Ontario lower drain-tunnel is now more than 4 miles long.

The formations are sedimentary, dipping about 30° NW. and N., and traversed by many fissures, dikes, and faults. The principal ore-bodies are found associated either with the contact of the basal Ontario quartzite and the overlying limes, or on a quartzite stratum within the limes, or in the fissures, or along some of the porphyry dikes. The district covers the intersection of the Uintah and Wasatch ranges, which explains the existence of large folds and faults, and fissures of great length, width, and depth.

In a general way, the geology of the district consists of a basal quartzite dome, which has been exposed by erosion at the Ontario property. The Ontario fissure lies entirely within this formation. Flanking this dome on the west, north, and east lie

the Park City limestones, dipping about 30° towards the NW., N., and NE., respectively, with an average thickness of 600 ft. This formation is Upper Carboniferous, and is the home of the most valuable deposits. Above the Park City formation there is 1,100 ft. of Woodside shales. The importance of these impervious shales to the underlying limes is readily seen. Above the shales occurs the Thaynes Canyon lime formation, which is secondary in importance to the Park City limes as an ore-bearer. This formation was classified by King as Permo-Carboniferous. It has a thickness of 1,200 feet.

The description of the ores given by Mr. Boutwell in 1903 still holds good, though we are now depending more on the low-grade milling-ores for our production than upon the high grades of earlier days. The ores are essentially argentiferous lead-ores with accessory gold and copper and a siliceous gangue. The values in the sulphide ore lie in galena, tetrahedrite, and pyrite; and in the oxidized ore in cerussite, anglesite, azurite, malachite, and complete oxidation-products. Silver has also been found in its native state. Zinc is a common associate in fissure-ore. Barite and fluorite occur sparingly. An average high grade carries about 60 oz. of silver, 40 per cent. of lead, 0.25 oz. of gold, and 2.5 per cent. of copper. Ordinary crude shipping-ore will average 50 oz. of silver, 22 per cent. of lead, 0.08 oz. of gold, and 1.5 per cent. of copper. Some zinc occurs with these two classes of ores; but the smelters do not pay for it. The milling-ore will average from 10 oz. of silver, 4 per cent. of lead, and 10 per cent. of zinc up to the values of the crude shipping-ore.

The fissures were the main avenues, allowing great freedom to the mineral-bearing solutions as they ascended through the quartzites and limes to the surface. I believe that the ore-deposition of the district is genetically connected with the vast masses of intrusives existing at great depths. Both magmatic and meteoric waters probably played parts in redistributing the values. The porphyry dikes of the district are of two ages: the older being of the same age as the fissures and in some instances playing the same rôle in circulating values; the younger porphyry dikes were formed subsequent to the most active period of uplifting and mineralization, and are barren.

Replacement-deposits are found where the fissures cut the

limes. Deposits of the fissure type, in some instances, are not exhausted at a depth of 2,100 ft., while the replacement-deposits have been followed for more than 3,400 ft. on the dip of the lime-beds.

The ore-bearing solutions carried silver, lead, zinc, iron, and some gold, all in primary combinations. Under the influence of the oxidizing agencies, the zinc and iron gave way, differentially enriching the lead, which does not oxidize readily. That lead which does oxidize is the first to re-precipitate, thus enlarging the original lead-zone. Iron is then re-precipitated below the lead. Zinc re-precipitates last, and is found at the deeper levels.

While it is desirable that data of mining-costs should cover large tonnages, I have preferred in this instance to take a number of representative headings, giving a brief description of the conditions obtaining at each place, believing that the reader can form a better idea in this than in any other way, of the work done.

The data here given have been furnished by the operations of one of the large producing mines and are fairly representative for the district. Other properties may have enjoyed lower mining-costs, due to the opening of large homogeneous ore-bodies; but such conditions are more or less exceptional and temporary.

The total cost of development-work in the mines of the district ranges from $1.03 to $8.07 per ton of ore produced. The total cost of stoping ranges from $1.57 to $4.80 per ton. These high costs are due to the following conditions: (1) the irregularity of the ore-deposits in the limestone-beds, which must be followed for long distances in a manner which precludes economy of development; (2) great vertical and horizontal distances from the main haulage-ways to the surface; (3) the necessity of separating at the mines the smelting-ore, the milling-ore, and the waste.

In mining the high-grade carbonate ores, it is our custom to "stay with" the ore in its wanderings along the beds, and to take it all out as we go. This makes it possible that a very valuable deposit may have but one small face exposed at any one time,—which is perplexing to examining engineers who wish to "block out" a tonnage. In the operation of the larger

mines there have been long periods during which there was practically no ore in sight, yet the usual production and profit have been maintained. It is rather an exception when any considerable tonnage can be "blocked out." Such a condition occurs sometimes in deposits of the fissure type.

For purposes of comparison, the cost of labor, timber, and explosives is given.

Statements of Cost.

Labor-Costs (Eight-Hour Shifts).

Shift-bosses,	$5 00
Machine-men,	3.25
Miners,	3.00
Muckers,	3.00
Trammers,	3.00
Timber-men,	3.50
Timber-men's helpers,	3.00
Station-tenders,	3.50
Track- and pipe-men,	3 25
Top car-men,	3.00
Skinners,	3.00
Donkey-engineers,	3.00
Sample-men,	3.50
Powder-men,	3.50

Timber.

Oregon Size	Per 1,000 Board Feet.	Native. Size	Per 1,000 Board Feet
12 by 12 in.,	$23.50	10 by 10 in.,	$16.50
10 by 10 in.,	23.50	9 by 9 in.,	16.50
8 by 8 in.,	22.05	8 by 8 in.,	16.50
6 by 8 in.,	23.30	6 by 6 in.,	16.50
3 by 12 in.,	22.25	2 by 12 in.,	20.00
3 by 10 in.,	22.25	2 by 10 in.,	20.00
2 by 12 in.,	20.30	3 by 8 in.,	18.50
1 by 12 in.,	20.30	3 by 6 in.,	18.50
2 by 4 in.,	20.55	2 by 12 in.,	18.50
3 by 6 in.,	20.80	2 by 10 in.,	18.50
2 by 10 in.,	20.30	2 by 8 in.,	18.50
2 by 8 in.,	20.30		

Poles (Native).

	Cents per Linear Foot.
3 in.,	3.25
4 in.,	4.5
6 in.,	4.5
7 in.,	5.5
8 in.,	6.5
9 in.,	7.5
10 in.,	8.5
11 in.,	9.5
12 in.,	10
14 in.,	12

Wedges, $1 per 100.
Ladders, 8 cents per foot.
Coal, $3.70 per ton at the boilers.
Candles, $0.0092 apiece.

Powder.

1,000 sticks of $\frac{7}{8}$ powder weigh 281.0 lb. and cost $38.34.
1,000 sticks of $1\frac{1}{8}$ powder weigh 540.0 lb. and cost $64.40.
4 X caps cost $5.80 per 1,000.
5 X caps cost $7.00 per 1,000.
1,000 ft. of Blue Label fuse costs $3.22.
1,000 ft. of Victor fuse costs $4.12.
1,000 ft. of Eagle fuse costs $5.00.

Drifting.

Record covers a 12 days' run. Size of drift, 11 by 7 ft. Driven on contact-vein in quartzite with a heavy black lime hanging-wall. Good air. Timbered. Dry. In ore. Mule-tram, 1,750 ft. Hoisting in cars through vertical shaft.

Large Machines Used; 22 Machine-Shifts Worked.

	Amount	Cost Per Foot.	Cost. Per Ton.
Machine-men,	$138.12	$2.44	$0.39
Muckers,	127.50	2.26	0.36
Pipe- and track-men,	12.00	0.21	0.04
Timber-men,	31.50	0.56	0.09
Miscellaneous labor,	14.00	0.25	0.04
Labor-cost,	$323.12	$5 72	$0.92
Cost of operating machines,	$88.00	$1.55	$0.25
Explosives,	31.55	0.56	0.09
Lumber and timber,	72.12	1.28	0.21
Hoisting,	87.25	1.54	0.25
Supplies,	3.46	0.06	0.01
General expense,	22 62	0.40	0.07
Total cost,	$628.12	$11.11	$1.80

Cross-Cutting.

Record covers a 12 days' run. Size of heading, 7 by 7 ft. Three shifts per 24 hr. Two large ($3\frac{1}{8}$-in.) machines drill from the same column each shift. Three machine-men and two muckers work each shift. Each shift blasts. The heading was driven in hard quartzite dipping 25° in the direction of advance. Dry face. Good air. The material was waste, hand-trammed 1,000 ft. and hoisted in cars.

Large Machines Used; 66 Machine-Shifts Worked.

All labor was performed by contract at the rate of $7 per foot.

	Amount	Cost Per Foot.	Cost Per Ton
Labor-cost,	$651.00	$7.00	$1.89
Cost of operating machines,	264.00	2.84	0.77
Explosives,	229.06	2.46	0.66
Hoisting,	86.02	0.92	0.25
Supplies,	11.90	0.13	0 03
General expense,	51.02	0.55	0.15
Total cost,	$1,293.00	$13.90	$3.75

Cutting Station.

Dimensions of the station are 18 ft. wide, 7 ft. high and 55.5 ft. long. Driven in hard quartzite. Dry. Good air. No tramming. Hoisting in cars.

Large Machines Used; 31 Machine-Shifts Worked.

The cost of machine-men, muckers, and timber-men is not segregated, but appears in a lump sum. General expense is included in the labor-cost.

	Amount.	Cost Per Foot	Cost Per Ton.
Labor-cost,	$1,140.25	$20.53	$2.10
Cost of operating machines,	124.00	2.25	0.23
Explosives,	133 88	2.42	0.24
Lumber and timber,	127.09	2.28	0.23
Hoisting,	136.50	2.46	0.25
Supplies,	21.11	0.38	0.04
Total cost,	$1,682.83	$30.32	$3.09

In driving this station one of the machine-men was put in charge of the work and held responsible, thus relieving the regular shift-bosses of a trip to the station and down the shaft.

Driving Raise.

Record covers a period of 20 days. Raise on contact-vein in the quartzite. Dimensions of raise, 17.5 by 5.5 ft. Lime hanging-wall. Good drilling and breaking. Dry. Good air. Driven on ore. Mule-tram of 1,500 ft. and hoisted.

Small Machines Used; 25 Machine-Shifts Worked.

	Amount.	Cost Per Foot.	Cost Per Ton.
Machine-men,	$81.25	$1.31	$0.16
Muckers,	58.50	0.95	0.11
Timber-men,	68.25	1.10	0.13
Pipe- and track-men,	3.00	0.05	0.01
Labor-cost,	$211.00	$3.41	$0.41
Cost of operating machines,	$50.00	$0.81	$0.10
Explosives,	33.55	0.54	0.07
Lumber and timber,	81.48	1.32	0.16
Hoisting,	127.25	2.06	0.25
Supplies,	4.80	0.08	0.01
General expense,	16 28	0.26	0.03
Total cost,	$524.36	$8.48	$1.03

Winzing.

Record covers a 7 days' run. Dimensions of the winze, 11 by 7.5 ft. Sunk on contact-vein in quartzite with a lime hanging-wall. Good drilling and breaking. Dry. Good air. Sunk on ore. Depth of winze at the time of gathering data, 110 ft. The material was hoisted 110 ft. out of the winze, mule-trammed 1,200 ft., incline-hoisted 200 ft., and hoisted up the main shaft in cars.

Large Machines Used; 4 Machine-Shifts Worked.

	Amount.	Cost Per Foot.	Cost Per Ton.
Machine-men,	$28.00	$3.73	$0.95
Muckers,	60.00	8.00	2.04
Timber-men,	50.75	6.77	1.70
Miscellaneous labor,	19.50	2.60	0.67
Labor-cost,	$158.25	$21.10	$5.36
Cost of operating machines,	$16.00	$2.13	$0.54
Explosives,	8.84	1.18	0.30
Lumber and timber,	9.76	1.30	0.33
Hoisting,	20.65	2.75	0.70
Supplies,	4 39	0.59	0.15
General expense,	1.26	0.17	0.04
Total cost,	$219.15	$29.22	$7.42

Sinking Vertical Shaft.

Dimensions of shaft, 17 by 6.5 ft. over all. Record covers a period of 30 days. Sunk through hard quartzite. Continuous pumping for a distance of 300 ft. Depth of shaft, 1,700 ft. Rock hoisted to the nearest level by bucket, transferred, and hoisted to the surface by cage.

Large Machines Used; 12 Machine-Shifts Worked.

The labor-cost of machine-men, muckers, timber-men and bosses is not segregated, but appears in a lump sum.

	Amount	Cost Per Foot.	Cost Per Ton.
Labor-cost,	$1,140.25	$42.23	$5.70
Cost of operating machines,	48.00	1.78	0.24
Explosives,	86.33	3.20	0.43
Lumber and timber,	105.20	3.90	0.53
Hoisting,	100.00	3.69	0.50
Supplies,	42.00	1.56	0 21
General expense,	91.22	3.38	0.46
Total cost,	$1,613.00	$59.74	$8.07

Stoping.

Stopes Nos. 1, 2, and 3 have an average width of 10 ft. The ore occurs in quartzite with some lime. Drills and breaks easily. Square-set timbering used. No sorting necessary, as all the rock goes for mill-ore. No stope-filling placed at the time these costs were compiled. Rather soft hanging-wall. Distance from the shaft, about 1,500 ft. All the labor-costs are taken as shown by the pay-roll. The cost of machines includes everything that can be charged to the operation of the machine, such as sharpening steel, air, repairs, etc. The cost of explosives includes the total cost of all powder, fuse, and caps, as given in the table of powder-costs. The cost of hoisting includes everything which can be charged to hoisting, such as steam, hoisting-engineers, shaft-repairs, etc. This item is high because the rock is still handled in cars through the shaft. Heavy hoisting-charges do not obtain throughout the district, as a rule. Lumber and timber are charged with the actual cost, delivered at the shaft. The cost of supplies covers all supplies going into the stopes, including candles. The general expense includes the wages of bosses, the cost of assaying, surveying, and all underground work of a general nature affecting the cost of stoping, such as powder-men, top car-men, etc.

Small Machines Used. Machine-Shifts Worked: Stope No. 1, 18; *Stope No.* 2, 73; *Stope No.* 3, 8.

	Stope No 1	Stope No. 2	Stope No. 3
Machine-men,	$58.50	$237.25	$26.00
Muckers,	12.00	255.00	27.00
Pipe- and track-men,	9.00	60.00	21.00
Timber-men,	10 50	106.75	31.50
Miscellaneous,	0.00	52.00	6.00
Total labor,	$90 00	$711.00	$111.50
Cost of operating machines,	$36.00	$146.00	$16.00
Explosives,	16.50	84.20	14.36
Lumber and timber,	114.16	493.11	160.36
Hoisting,	33.75	401 62	39.00
Supplies,	4.09	10.72	9 88
General expense,	6.55	56 00	8.00
Total cost,	$301.05	$1,902.65	$359.10
Tons of ore,	121.5	1,606.5	156.0
Tons of waste hoisted,	13.5	0.0	0.0
Total tons,	135.0	1,606.5	156.0
Cost per ton,	$2.23	$1.19	$2.30

Stope No. 4.—The average width of this stope is 16 ft., othe wise the conditions are practically the same as those in stop Nos. 1, 2, and 3. Distance from the shaft, 1,700 feet.

Small Machines Used; 15 *Machine-Shifts Worked.*

	Amount
Machine-men,	$48.75
Muckers,	52.50
Timber-men,	32.50
Total labor,	$133.75
Cost of operating machines,	$30.00
Explosives,	15.11
Lumber and timber,	112.00
Hoisting,	56.30
Supplies,	9.52
General expense,	24.07
Total cost,	$380.75
Tons of ore,	232.2
Tons of waste hoisted,	9.0
Total tons,	241.2
Cost per ton,	$1.57

Stope No. 5.—This stope has been chosen because it represents a type frequent in the Park City district. The width varies from 4 to 11 ft. The timbering is partly stull and partly square-set. The stope is 225 ft. above the level and 4,500 ft. from the shaft. Very wet, gum clothes being necessary for the most part. The ore is being followed up and along the beds of lime, necessitating a triple handling of the ore before it reaches the level. The ore is mule-trammed to a point near the shaft, dropped to the tunnel-level by chute, and hauled to the railroad.

Small Machines Used, 121 *Machine-Shifts Worked.*

	Amount.
Machine-men,	$418.00
Hand miners,	276.25
Muckers,	290.50
Pipe- and track-men,	10.50
Timber-men,	225.00
Miscellaneous labor,	67.00
Total labor,	$1,287.25
Cost of operating machines,	$242.00
Explosives,	59.64
Lumber and timber,	90.06
Hoisting,	98.52
Supplies,	25.65
General expense,	90.10
Total cost,	$1,893.22
Tons of ore,	390.6
Tons of waste hoisted,	3.5
Total tons,	394.1
Cost per ton,	$4.80

Geology of the Cobalt District, Ontario, Canada.

BY REGINALD E. HORE,* HOUGHTON, MICH.

(Wilkes-Barre Meeting, June, 1911)

I. INTRODUCTION.

SINCE the discovery of silver at Cobalt, Ontario, in August, 1903, more than 100,000,000 oz. of silver have been produced by the mines in the Nipissing district, and there is reason to believe that at least as much more will be produced in the next five years. The estimated value of the aggregate output of ore to the end of 1910 is $48,327,280. The ore yielded 93,977,833, oz. of silver and was mined at a net profit of about $26,000,000. For 1910 the production was 30,558,825 oz., valued at $15,436,894, and yielding a profit of about $9,000,000. The details of the production of the Cobalt district for the years 1904 to 1909, inclusive, as reported by the Ontario Bureau of Mines, are given in Table I.

TABLE I.—*Silver-Production of the Cobalt District*, 1904 *to* 1909.

SMELTING-ORE.

	Production.	Value	Silver-Content.	Value Per Ton.
	Ounces.		Oz. Per Ton.	
1904......	206,875	$111,887	1,309	$708
1905......	2,451,356	1,360,503	1,143	634
1906......	5,401,766	3,667,551	1,013	687
1907 .. .	10,023,311	6,155,391	677	416
1908......	18,022,480	8,468,293	736	363
1909......	22,436,355	10,809,872	809	389
		CONCENTRATES.		
1908......	1,415,395	$665,085	1,240	585
1909......	3,461,470	1,651,704	1,174	559

The shipments from Cobalt are in the form of smelting-ore, concentrates from milling-ore, and bullion. The smelting-ore is largely high grade, averaging about 3,000 oz. per ton. A smaller return is from what is called low-grade ore and which

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averages about 200 oz. per ton. The ore milled in the camp averages about 30 oz. per ton. In 1910 the bullion shipped contained about 940,000 oz. of silver, and a much larger amount will be sent out in 1911.

Nearly all of the ore has been obtained from mines located within 3 miles of the original discovery; but silver- and cobalt-ores have been found in widely-separated areas, and there are now well-established camps at Gowganda[1] and South Lorrain. Neither of these camps can rival Cobalt, yet each one, while operating under adverse conditions, has shipped several carloads of rich ore and has developed considerable concentrating-ore. One mine in Casey township, north of Lake Temiskaming, makes occasional small shipments. The camps at Elk lake and at Maple mountain have attracted considerable attention, though the ore-deposits so far discovered are small and irregular.

These camps, and several others in which similar but non-productive deposits have been found, are in the district of Nipissing. They lie within a broad belt of Huronian rocks, the southern boundary of which stretches from Georgian bay, NE. to Lake Temiskaming and Quebec Province. There is thus a large field, about 80 miles square, in Nipissing in which numerous discoveries of native silver and cobalt arsenides have been made, and in which more will doubtless be made as exploration is continued. Immediately north of Gowganda lies the newly-discovered Porcupine gold-field. I have described the silver-fields in a general way in my paper.[2] The present paper relates chiefly to the geological features of the Nipissing district. A summary of recent developments at Cobalt has been published elsewhere.[3]

Fig. 1 is a sketch-map of the Cobalt district, and Fig. 2 is a view of the town of Cobalt from Nipissing hill. Fig. 3 illustrates the method of prospecting by trenches at the Nipissing mine, the Coniagas mine appearing in the background of the view towards the right-hand side, and the Buffalo mine

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[1] Silver Deposits of Gowganda District, Ontario, *Mining World*, vol. xxxii., No. 24, pp. 1171 to 1173 (June 11, 1910).

[2] The Silver Fields of Nipissing, presented at the Toronto meeting of the Canadian Mining Institute, in 1910; not yet published.

[3] *Engineering and Mining Journal*, vol. xci., No. 14, pp. 717 to 718 (Apr. 8, 1911).

at the left. Fig. 4 gives a nearer view of the plants at the Coniagas and Trethewey mines.

II. General Geology.

The district here described is underlain for the greater part by rocks of four distinct series, all believed to be pre-Cambrian. The oldest is a complex of much metamorphosed igneous and sedimentary rocks, designated by the name Keewatin. Intrusive into these is a series of siliceous, distinctly-grained igneous rocks, called Laurentian. Lying unconformably on both of these formations is the sedimentary series to which Logan gave the name Huronian. Intrusive into all of these are masses of diabase, here referred to the Keweenawan. The Laurentian and Keewatin together comprise the Archæan, and the Huronian and Keweenawan make up the Algonkian. There is, NW. from Lake Temiskaming, a series of fossiliferous sediments, chiefly limestone, which lies unconformably on the Algonkian, and which has been correlated with the Niagara of New York State.

Rich silver-ore has been mined from the Huronian sediments, from the Keewatin complex, and from the Keweenawan diabase; but none from the Laurentian rocks. Probably 90 per cent. of the silver has been taken from veins in Huronian sediments.

The geological section of the Cobalt district is outlined in Table II. The chief divisions were noted by Sir William E. Logan and Robert Bell in the early reports of the Canadian Geological Survey. The subdivisions were made by Dr. A. E. Barlow,[4] Dr. W. G. Miller,[5] and others as the result of more detailed mapping for the Dominion and Provincial governments. I offer the table in this form after having had numerous opportunities of observing the structural relations and examining microscopically several hundred rock-sections. Free use has been made of the literature bearing on the geology of the district and I append a list to this paper.

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[4] Report on the Geology and Natural Resources of the Area Included by the Nipissing and Temiscaming Map Sheets, *Geological Survey of Canada*, New Series, vol. x., pt. I., 303 pp. (1897). The Temagami District, *Summary Report of the Geological Survey Department of Canada*, pp. 120 to 133 (1903).

[5] Cobalt-Nickel Arsenides and Silver-Deposits of Temiskaming, *Fourteenth Report, Ontario Bureau of Mines*, pt. II., 66 pp. (1905); *Sixteenth Report*, pt. II., pp. 1 to 116 (1907).

TABLE II.—*Rocks of the Nipissing Silver-Fields.*

1. CENOZOIC:

Recent..............Clay, marl, peat.

Pleistocene.........(1) Coarse unstratified material—sand, gravel, boulders.
(2) Stratified clay with some sand.

Great unconformity.

2. PALÆOZOIC.

Silurian............Gray limestone with some interbedded greenish shales, and at the base an arenaceous conglomerate.
Correlated with Niagara of New York State.

Great unconformity.

3. ALGONKIAN:

Keweenawan......Igneous intrusives only. Chiefly quartz-diabase and quartz-gabbro with acid differentiation-products. Some olivine-diabase and diabase-porphyrite dikes.

Igneous contact.

Huronian..........Sedimentary rocks only
(a) An upper series. Probably equivalent to Middle Huronian of Lake Superior district. Chiefly feldspathic quartzite with some conglomerate.

Slight unconformity.

(b) A lower series. Probably equivalent to Lower Huronian of Lake Superior district. Chiefly graywacke, shale, conglomerate, and feldspathic quartzite. The conglomerate pebbles are mostly of holocrystalline igneous rocks, the matrix graywacke and gray shale. The rocks are seldom schistose except as the result of contact metamorphism.

Great unconformity.

4. ARCHÆAN:

Laurentian........Igneous intrusives only. Holocrystalline light-colored siliceous rocks. Chiefly granites, diorites, syenites, and gneisses.

Igneous contact.

Keewatin...........Igneous and sedimentary rocks. All much metamorphosed and many schistose. The relative age of the igneous and sedimentary rocks is doubtful. The igneous rocks are chiefly of extrusive types.

Extrusives......(1) Dark-colored basic rocks—basalts—mostly with composition and texture of altered diabases.
(2) Light-colored siliceous rocks—felsite-porphyries—mostly quartz-porphyries which have been altered to sericite-schists.

Intrusives.......(1) Basic rocks, mostly diabase and gabbro.
(2) Siliceous rocks, mostly quartz-porphyries and porphyrites.

Sediments.......(1) The iron-formation, chert, jaspilite, carbonates, slates, and green schists.
(2) Fragmental volcanic rocks—a gray felsite agglomerate.

III. Petrology.

1. *Keewatin Formations.*

The Keewatin rocks are of very numerous types; some igneous, some sedimentary, and all much metamorphosed. None are of great areal extent. The igneous are more widespread than those that are believed to be of sedimentary origin.

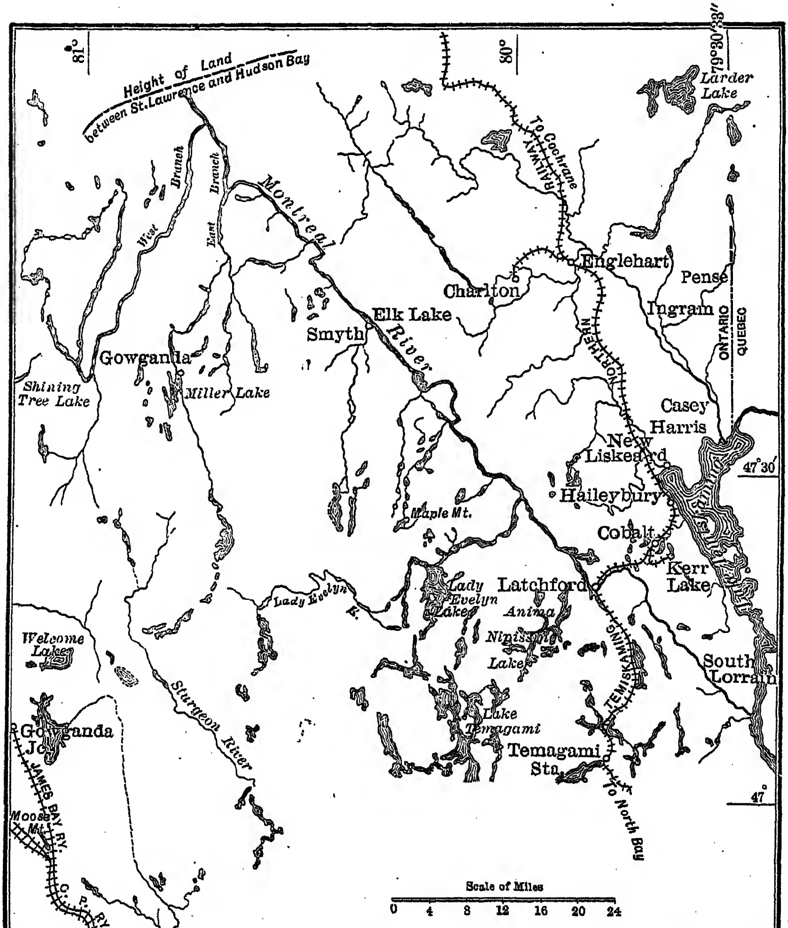

Fig. 1.—Sketch-Map of the Cobalt District, Ontario.

The igneous rocks are for the most part fine-grained types, varying in color from dark greenish black to very light gray, and in composition from basic to highly siliceous. Medium- to coarse-grained dark-colored masses intrude those of finer

grain, and holocrystalline light-colored siliceous rocks are typically absent.

The most widespread of the Keewatin rocks are the fine-grained dark-colored basaltic types, which resemble closely the Mona schists of Michigan. These rocks are always much altered, and from the color of characteristic decomposition-products are conveniently referred to as greenstones. Most frequently the chief original minerals found in them are soda-lime feldspars, pyroxene, hornblende, and iron-ores. Ophitic textures are found in many of the specimens examined, and rocks of the composition and texture of altered diabase are especially prominent. There are also dark-colored rocks in which there is a marked foliation and to which the terms chlorite- and hornblende-schist are applicable. The light-colored volcanics are much less abundant, and are not found in all the localities in which the Keewatin rocks are well developed. As a rule their original character is obscured and they have a schistose structure. A common type is yellowish to greenish sericite-schist stained by decomposition-products of pyrite.

Dike rocks of various types are found intruding the volcanics. Among dark-colored ones, olivine-diabase, diabase, and lamprophyres are common. Quartz-porphyries are prominent among the light-colored varieties.

Fragmental igneous rocks are comparatively rare in this district. They are usually of rather light color, gray to greenish, and of intermediate composition, and resemble the Kitchi schists of the Marquette iron range.

The rocks thought to be of sedimentary origin are cherts, carbonates, slates, schists, and jaspilites. They constitute what is commonly known as the "iron-formation." The surface-exposure of some iron-formations suggests truncated sharply-folded synclines of sediments which were originally ferruginous carbonates, cherts, and shales. The metamorphic rocks produced from these are now inclosed by igneous rocks, on which they may have been deposited. It has been remarked that most of the Keewatin rocks are of volcanic types, and that the fragmental ones have characters similar to those produced by water-action. It is possible, therefore, that much of the igneous material was emitted by submarine volcanoes, and that the sediments are of practically contemporaneous origin.

There are several rich veins of silver in Keewatin rocks at Cobalt. Most of the recently-discovered gold-quartz deposits at Porcupine, 50 miles north of Gowganda, are in Keewatin rocks—chiefly in schists impregnated with carbonates. I have recently described [6] these gold-deposits.

2. *Laurentian Formation.*

The rocks of this series are for the most part holocrystalline light-colored siliceous types. They are medium- and coarse-grained granites, diorites, and syenites. Quartzose varieties are especially prominent, and red granites are the most common members of the group. Some of the granites are practically free from dark-colored minerals, while others are characterized by biotite and hornblende. The syenites usually show green hornblende. In many parts of the district gneissoid structure is not specially prominent, thus differing markedly from the rocks of the original Laurentian area. Wherever these rocks have been found in contact with those described above as Keewatin, they intrude the latter. So far as I am aware, no rocks of this type in the silver-field have been found intrusive into the sediments described below as Huronian. In numerous instances I have found Huronian conglomerates which lie uncomformably on granites and syenites referred to the Laurentian. No deposits of economic importance have been found in Laurentian rocks in Nipissing.

3. *Huronian Formation.*

All of the rocks of the Huronian series are of sedimentary types. The lower beds are chiefly conglomerate, shale, feldspathic quartzite, and graywacke. Slaty cleavage, found in many instances, is only locally developed, and there are few large areas of true slates. Fig. 5 is a view of a typical outcrop of Huronian sediments in the Temagami Reserve; the well-stratified shaly graywacke is overlain by the massive gray-

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[6] *Canadian Mining Journal*, vol xxxi., No. 20, pp. 617 to 622 (Oct. 15, 1910); No. 21, pp. 649 to 656 (Nov. 1, 1910); vol xxxii., No. 3, pp. 82 to 86 (Feb. 1, 1911). *Engineering and Mining Journal*, vol. xc, No 27, pp. 1296 to 1298 (Dec. 31, 1910). *Mining and Scientific Press*, vol. ci., No. 22, pp. 705 to 706 (Nov 26, 1910); vol. cii., No. 17, pp. 588 to 591 (Apr. 29, 1911). Also, Quebec Meeting, Canadian Mining Institute, March, 1911.

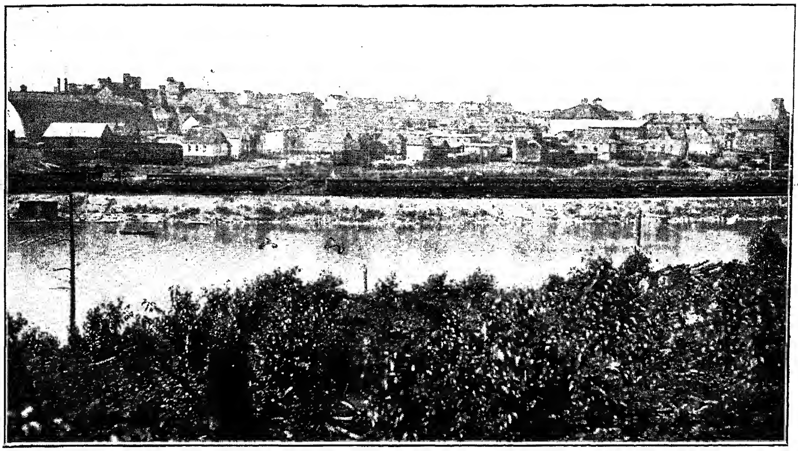


Fig. 2.—A View of Cobalt from Nipissing Hill.

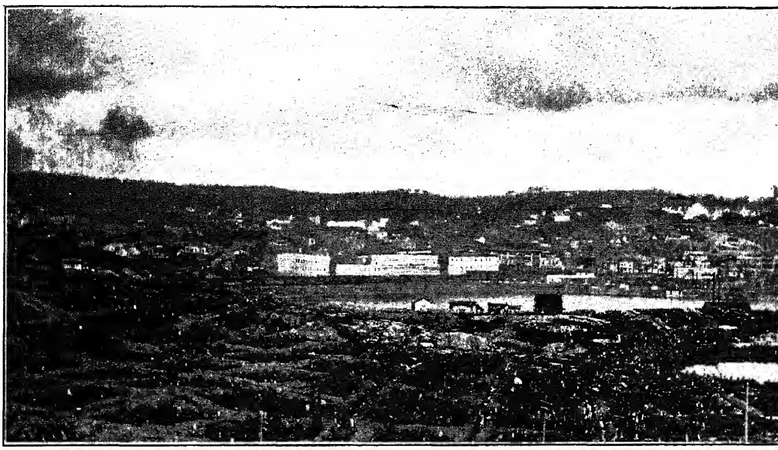


Fig. 3.—Cobalt from Nipissing Hill, Looking West, Showing Method of Prospecting by Trenches. Coniagas Mine in Right Background.

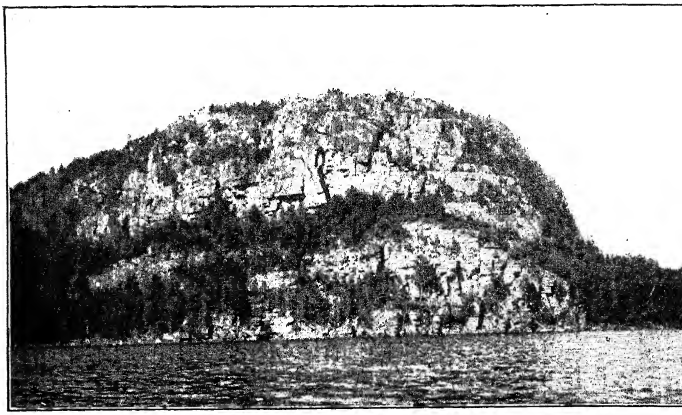


FIG. 5.—A TYPICAL OUTCROP OF HURONIAN SEDIMENTS, TEMAGAMI RESERVE. WELL-STRATIFIED GRAYWACKE OVERLAIN BY MASSIVE GRAYWACKE CONGLOMERATE.

FIG. 6.—HURONIAN CONGLOMERATE, CONIAGAS MINE, COBALT.

FIG. 7.—WEATHERED SURFACE OF HURONIAN CONGLOMERATE, NIPISSING MINE, COBALT.

FIG. 8.—SILVER-SMALTITE VEIN, 9 IN. WIDE, LAWSON MINE, COBALT.

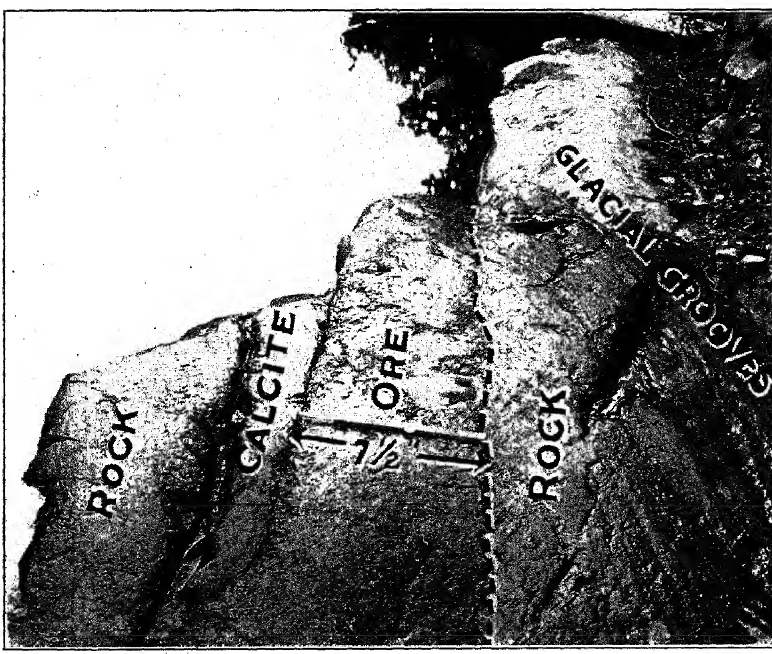
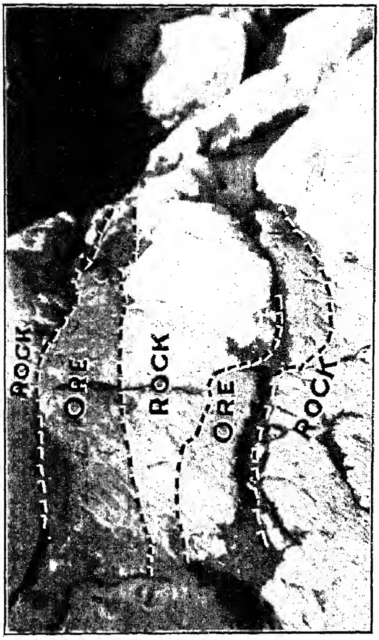


FIG. 9.—RICH ORE SHOWING AT THE SURFACE ON THE LAWSON PROPERTY OF LA ROSE MINES. SHOWS HOW THE READILY-WEATHERING SMALTITE HAS BEEN PRESERVED SINCE GLACIAL TIMES BY A FEW FEET OF DRIFT.



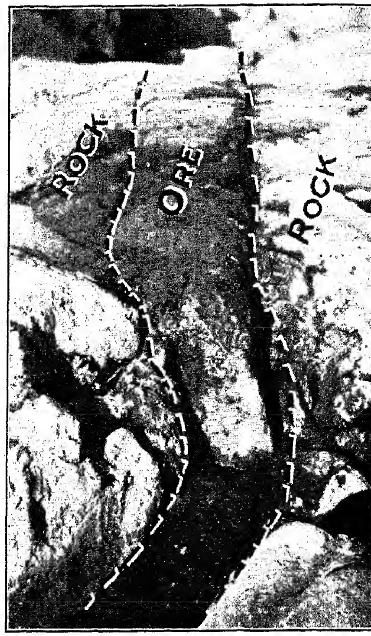


FIG. 10.—SILVER-SMALTITE VEINS, COBALT.

FIG. 11.—SMALTITE-SILVER VEIN, COBALT.

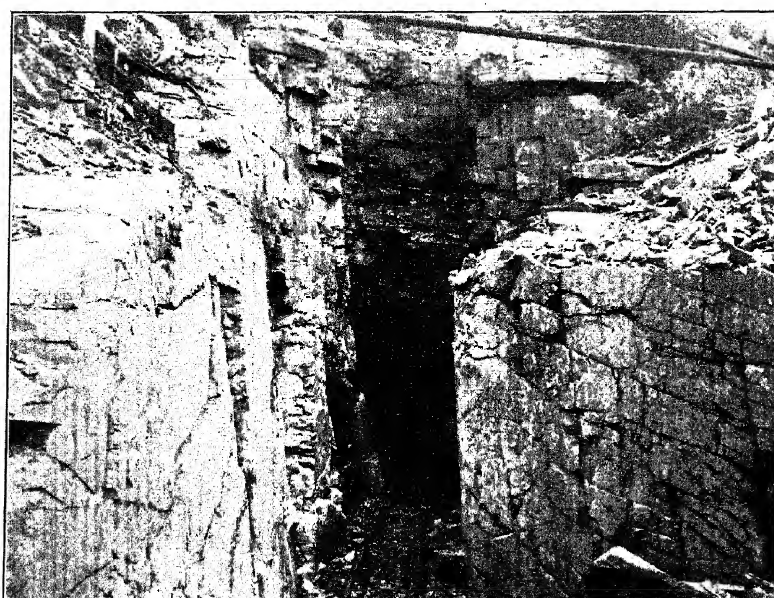


FIG. 12.—ADIT ON NARROW VEIN, LA ROSE MINE. SHOWS JOINTING AND BEDDING IN GRAYWACKE.

wacke conglomerate. Fig. 6 illustrates the conglomerate which occurs at the Coniagas mine, and Fig. 7 the weathered surface of an outcrop at the Nipissing mine.

The usual succession is, a basal conglomerate with a dark-colored hardened mudstone matrix, grading into graywacke and shale with no large pebbles. Above the shales are feldspathic quartzites, and these are overlain by a massive conglomerate. The thickness of the beds varies greatly, partly owing to the very irregular contour of the underlying rocks, and to marked differences in erosion. At Cobalt the series is rarely 300 ft. thick; but Dr. Parks describes a 550-ft. vertical section of similar rocks at Chaminiss hill, east of Larder lake.

A younger series of sediments consisting largely of feldspathic quartzite overlies those described above. In most instances where exposures showing the relations have been discovered, there is a gradual transition from the lower to the upper series. In other cases there is a definite line of demarcation between the two series. In a few instances there is a discontinuity of deposition expressed by a basal conglomerate.

1. *Conglomerate.*—The conglomerate of the lower series is a very peculiar type of rock, and it is difficult to interpret its mode of formation. The distribution of the pebbles is very irregular, and in some parts there are rounded boulders several inches in diameter, scattered at wide intervals through fine-grained graywacke and shale. The pebbles are of many types, the most conspicuous being red and gray granites, similar to the rocks of the Laurentian. The darker-colored boulders are in part granitoid types; but frequently are fine-grained rocks similar to the more massive members of the Keewatin series. Boulders similar to the more readily disintegrated schists and slates of the Keewatin series are only conspicuous in portions of the conglomerate in the immediate vicinity of the latter. There is a marked similarity in the types of boulders in the conglomerate in all parts of the area; but near the base of the beds there is an unusually high percentage of types similar to those other rocks which are immediately adjacent.

With the exception of a few angular fragments at the base, the boulders are generally well rounded, as though water-worn; others are subangular. The arrangement of the boulders and

the character of the matrix suggest glacial débris. See Dr. A. P. Coleman's paper, The Lower Huronian Ice Age,[7] and my paper, The Glacial Origin of Huronian Rocks of Nipissing.[8]

2. *Shales.*—Intimately associated with the conglomerates are hard, distinctly-bedded shales, which for the most part are gray in color, and less often greenish black. Occasionally these shales are interbanded with layers of purple, green, and pale gray colors. All are composed partly of minute indeterminable particles. The chief recognizable minerals are quartz and decomposed feldspar, minute scales of chlorite and sericite, and small grains of epidote, biotite, and iron-ores.

3. *Graywacke.*—There are some rocks, closely allied to the shales and arkoses, to which the term "graywacke" is applied. The chief recognizable constituents are feldspar, quartz, a dark chloritic mineral, and a pale-colored mica. Less abundant are small particles of iron-ore and epidote, while biotite, pyroxene, and amphibole are rare. With the minerals are angular and rounded rock-particles of various sizes. Rock of this type in some instances is found in massive beds of uniform character, and similar material forms the matrix of much of the boulder conglomerate.

4. *Quartzite and Arkose.*—The most widespread and thickest beds of Huronian rocks are quartzites and arkoses. The quartzites are in most instances feldspathic and grade insensibly into typical arkoses. They are usually very massive, fine to medium grained, and not unlike light-colored granites in appearance.

Most of the rich deposits at Cobalt are in Huronian rocks, and especially in the conglomerate. In no part of the district have ores been found in the upper series of the Huronian.

4. *Keweenawan Formation.*

The igneous masses referred to this series are of types having for the most part the composition and texture of diabases,[9] the most abundant being medium-grained gray quartz-diabase. Coarser-grained types are in part typical diabase, and less often of the texture of gabbros. Some red-colored and quartzose

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[7] *Journal of Geology*, vol. xvi., No. 2, pp. 149 to 158 (Feb.–Mar., 1908).

[8] *Journal of Geology*, vol. xviii., No. 5, pp. 459 to 467 (July–Aug., 1910).

[9] Diabase of the Cobalt District, *Journal of Geology*, vol. xviii., No. 3, pp. 271 to 278 (Apr.–May, 1910).

masses are closely connected in composition and origin with the gray diabase, while aplitic veins[10] of soda granite are probably siliceous differentiation-products. There are also some dikes of olivine-diabase and of diabase-porphyrite.

In none of these masses do the structures or textures indicate volcanic origin, thus differing markedly from the copper-bearing rocks with which they are here correlated. I regard these masses as the deep-seated equivalents of the Keweenawan volcanics of Lake Superior.

The Diabase Masses.—There is great variety in the size and shape of the exposed masses. Many are of decided sheet-like form. The sills vary in thickness from 100 to 500 ft., and in extent can seldom be traced continuously for more than a few miles.

In many cases the diabase conforms to, and has apparently its shape determined by, the bedding-planes of underlying shales. The shales usually dip slightly towards the diabase, suggesting that collapse followed the sealing of the feeding-channels. Other masses show in places decided stock-like contacts, and many of the smaller outcrops are dikes.

The greater portion of the diabase masses is of dark-gray color and of medium grain. The specific gravity is about 3. The rock is composed chiefly of grayish or greenish soda-lime feldspars, set in dull brown pyroxenes. Biotite and black iron oxides are generally also visible. Quartz is frequently present, though often in small quantity interwoven with feldspar, and then not always visible to the naked eye.

In some specimens there is a decided pink color, due to the presence of pink sodic feldspar. In these portions quartz is more prominent, and grains of chalcopyrite and pyrite are frequently visible.

While in some instances the texture is that of gabbro, the diabasic character is generally developed in more or less degree, and the term "diabase" has, therefore, been used in this article as a designation for the rock-masses. For some minor portions of the masses, the term "gabbro" should be applied. Other small portions are albite-granites and quartz-gabbro. The red portions, like the Lake Superior "red rocks,"

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[10] Differentiation Products in Quartz Diabase Masses of the Silver Fields of Nipissing, *Economic Geology*, vol. vi., No. 1, pp. 51 to 59 (Jan.–Feb., 1911).

usually show the intergrowth of quartz and feldspars characteristic of micropegmatites.

IV. Structure of the Deposits.[11]

The ore-bodies are all fissure-fillings, and the veins are but a few inches wide. Surface-exposures of some silver-smaltite veins in the Cobalt district are shown in Figs. 8, 9, 10, and 11. The jointing and bedding in the shaly graywacke is shown in Fig. 12, a view of the adit along a narrow vein at the La Rose mine. There is, with a few important exceptions, little evidence of extensive faulting; but in numerous cases slight, and nearly horizontal, displacements took place previous to and also subsequent to the deposition of the ore. At the south end of Cobalt lake a number of faults, one of which shows a vertical displacement of 400 ft., have been encountered.

In some veins, post-glacial weathering has resulted in the decomposition and removal of part of the filling. At the surface of rich veins silver nuggets have been found in dark earthy material partly made up of cobalt oxide. These decomposition-products are especially characteristic of veins not covered by a mantle of drift. In some veins thus protected fresh smaltite is found but a few inches below the surface, and some of the veins of rich ore still show the marks of the ice action. Fig. 9 shows such a vein at the Lawson mine.

Many of the fissures are nearly vertical, and most of the others are steeply inclined. In the Huronian sediments, the fissures, usually vertical and regular in direction, pass indifferently through boulders and matrix in the conglomerate. Fissures in the Keweenawan diabase are usually vertical or steeply inclined, while those in the Keewatin greenstones are usually inclined and irregular.

The fissures are almost all very small. Few productive veins have been followed 500 ft., and very few are known to persist horizontally more than 1,000 ft. The depth of the ore-filled fissures has been found in many instances to be from 100 to 200 ft., and in a few instances from 400 to 500 ft. Comparatively little ore has been taken from below the 300-ft. level, though a few deposits have been proved to greater depth. There are at present very few mines in which the workings are more

[11] For fuller discussion see *The Mining World*, vol. xxxiii., No. 17, pp. 747 to 751 (Oct. 22, 1910).

than 400 ft. deep; one in Keweenawan diabase, the others in Keewatin greenstones. Recent explorations below the 200-ft. level have resulted in the discovery of rich ore-bodies in some mines, and there is now more confidence that values will be found at depth.

V. Influence of the Country-Rock.

Veins in the Huronian conglomerate have yielded by far the greater part of the silver-product at Cobalt, and a vein in similar rocks at Miller lake, Gowganda, is the greatest single producer outside of Cobalt camp. Much rich ore has been mined from Keewatin rocks at Cobalt, but as a rule the values are less persistent than in the Huronian. Veins in the diabase are very numerous, but comparatively few are of importance. These include two highly-productive veins at Cobalt and a vein in South Lorrain in which a large tonnage of rich ore has been blocked out.

Some of the important fissures in the Huronian sediments terminate without reaching the underlying greenstones, others terminate at or near the contact, while still others continue down into the greenstones. Of the latter, most show a marked decrease in silver-content in the greenstones. The fissures also become irregular in direction, and not infrequently the veins break up into narrow stringers.

In passing from Huronian to Keewatin rocks, there is in nearly all cases a marked change in the character of the fissure, and in most cases also in the character of the fissure-filling. Invariably where there has been a marked change in values at the contacts it has proved a change for the worse. In some cases fissures have been followed down from Keewatin greenstones into the younger diabase intrusive, and at one mine a distinct improvement in silver-values was found in the diabase. The fissures have been also followed down from the diabase into greenstones and sediments. In one case a vein passed from diabase into underlying Huronian shales, with reported increase in silver-values in the shales.

From results of early workings, the operators have not had much faith in the Keewatin rocks; but during the past year some very rich ore-shoots have been found in these rocks at depth.

VI. The Ores.

1. *General Character.*—Native silver is the chief ore, and intimately associated with it are the cobalt-minerals, especially smaltite and erythrite (cobalt-bloom). While there are numerous cobaltiferous veins in which no native silver has been found, there are few, if any, native-silver-bearing veins in which no cobalt-minerals have been found. Generally with the smaltite is associated some niccolite. The gangue is calcite and dolomite. Quartz is comparatively rare in the producing veins.

The values are very irregularly distributed. In many of the best veins there are very frequent and sudden changes in content, so that of large samples taken a few feet apart in the ore-shoots, one may contain a few ounces and the next one several thousand ounces of silver. Shoots approaching uniform value throughout are seldom 100 ft. long. There are numerous calcite and dolomite veins that are barren.

The average silver-content of the ores is very high. Of the first two years' shipments to Ledoux & Co., 394 lots in all, 37.25 per cent. assayed more than 1,000 oz. per ton. The "nuggets" received by the same firm during those two years (1905–06) averaged 95 per cent. of silver. Ores carrying from 100 to 250 oz. per ton are generally spoken of as low grade, and the ore shipped to smelters seldom contains less than 60 oz. Many cars average 3,000 oz., and one car averaged about 9,000 oz., per ton. The concentrating-ores treated at Cobalt in 1908 averaged 32 oz. and yielded 28 oz. per ton, with a total of 1,415,395 oz. contained in 1,185 tons of concentrates. Table I. shows the total silver-production and its value, and the average content and value per ton. The shipments are classed as smelting-ores and concentrates. There are 14 concentrators but no smelters at Cobalt. Recently the Nipissing mine has found an economical method of reducing the high-grade ore and is now shipping bullion obtained by this process instead of ore.

2. *Cost of Production.*—The profit from the mining-operations at Cobalt has been remarkable. According to the report of Mr. Gibson,[12] Deputy Minister of Mines of Ontario, for the seven

[12] *Annual Report, Ontario Bureau of Mines* (1910).

years, 1904–1910, silver-ore valued at $48,327,280 was mined, and from this $21,802,150 was paid out in dividends and private companies made profits of more than $3,000,000. A study of the annual reports of the chief producers shows that the cost of producing ore is very high, yet the narrow veins are so rich that the margin of profit is in many cases 30 cents and in some cases 40 cents per ounce of silver. The cost of production varies considerably at different mines, but the leading shippers have taken out their ore at a cost of less than 20 cents per ounce of silver. Nipissing mine, the largest producer in the camp, reports total costs for 1909 to have been 16.39 cents per ounce and 14.72 cents for 1910. For 1910, Crown Reserve mine reports costs of 11.97 cents; Kerr Lake mine, 13.27 cents; and La Rose mine, 19.11 cents per ounce.

VII. Origin of the Deposits.

In the paper [13] presented before the Canadian Mining Institute in 1908, I discussed the origin of the ores, giving reasons for believing that the constituents were present in the diabase magma, that the Keewatin greenstones have aided in their deposition, and that the chief function of the sediments was affording suitable fissures for deposition.

It was stated that "we may expect to find similar ore deposits where the diabase sills are associated with Keewatin igneous rocks, and especially valuable deposits where Huronian sediments are also present."

Recent developments show that in the two most promising regions outside of Cobalt, namely, South Lorrain and Gowganda, the veins are in the vicinity of contacts of the diabase and altered greenstones. The most productive of these veins is one in Huronian sediments near the diabase and old greenstones. The theory outlined is therefore still considered tenable.

On the other hand, there have been found, west of Gowganda lake, important veins in a diabase ridge in the vicinity of which no old greenstones have been recognized. The same is

[13] Origin of Cobalt-Silver Ores of Northern Ontario. *Journal of the Canadian Mining Institute*, vol. xi., pp. 275 to 286 (1908); *Economic Geology*, vol. iii., No. 7, pp. 599 to 610 (Oct.-Nov., 1908).

true of silver-veins at Maple mountain. In these cases the influence of the intruded rocks is not apparent, and the ores occur in the irregularly-spaced jointing in the diabase itself. The genetic connection of the ores with the diabase is more or less evident in every camp.

VIII. Bibliography of Explorations in Nipissing Silver-Fields.

The following list of published maps and reports may prove useful:

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The Laurentian and Huronian Systems North of Lake Huron. With map. Robert Bell, *First Report, Ontario Bureau of Mines*, 1891, pp. 63 to 94.

Report of the Geology and Natural Resources of the Area Included by the Nipissing and Temiscaming Map Sheets. With maps. A. E. Barlow, *Geological Survey of Canada*, vol. x., N. S., Pt. I., 1897. 303 pp

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Boston Township Iron Range. W. G. Miller, *Fourteenth Report, Ontario Bureau of Mines*, Pt. I., 1905, pp. 261 to 268.

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The Ore Deposits of the Cobalt District. C. R. Van Hise, *Journal of the Canadian Mining Institute*, vol. x., pp. 45 to 61, 1907.

A Microscopic Examination of the Cobalt Nickel Arsenides and Silver Deposits of Temiscaming. A. Campbell and C. W. Knight, *Economic Geology*, vol. i., No. 8, pp. 767 to 776, 1906.

A Mineralized Area in the Townships of Casey and Harris. R. E. Hore, *Sixteenth Report, Ontario Bureau of Mines*, 1907, Pt. II., pp. 131 to 134. With map.

A Part of the Area South of Lake Wendigo. With map. R. E. Hore, *Sixteenth Report, Ontario Bureau of Mines*, 1907, Pt. II., pp. 135 to 137.

The Area West of Bay Lake on the Montreal River. With map. J. S. DeLury, *Sixteenth Report, Ontario Bureau of Mines*, 1907, Pt. II., pp. 138 to 146.

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Diabase and Granophyre of the Gowganda Lake District. N. L. Bowen, *Journal of Geology*, vol. xviii., No. 7, pp. 658 to 674, Oct.-Nov., 1910.

Geology of a Portion of Fabre Township. R. Harvie, Mines Branch, Department of Mines, Quebec, 1911.

MAPS.

The Ontario Bureau of Mines has published a number of geological maps covering portions of the Cobalt district. These maps, prepared by Dr. W. G. Miller, C. W. Knight, A. G. Burrows, and others, may be obtained, together with reports on the areas covered, by application to Thomas W. Gibson, Deputy Minister of Mines, Toronto.

The Canadian Geological Survey also has published maps prepared by Dr. A. E. Barlow, W. H. Collins, and others, which may be obtained by application to R. W. Brock, Director of the Geological Survey, Ottawa.

The Department of Lands, Forests, and Mines, Toronto, has published numerous maps of the Cobalt district and other parts of northern Ontario, which are distributed free to applicants.

Origin of Certain Bonanza Silver-Ores of the Arid Region.

BY CHARLES R. KEYES, DES MOINES, IOWA.

(Wilkes-Barre Meeting, June, 1911.)

CONTENTS.

I. INTRODUCTORY.

IN the dry regions of the globe many silver-deposits display certain remarkable features, which at the same time are so totally unlike anything met with among ore-bodies elsewhere that they have long presented exceptional difficulties, not only to a satisfactory explanation of their origin, but to economical milling. A most striking peculiarity of these ores is the haloid character of many of their bonanzas. To this fact more than to any other may be ascribed most of the unusual uncertainties of mining-operations in desert countries—the marvelous local richness of the ores under certain conditions and the often abrupt transition to utter poverty; the inadequacy of ordinary prospecting customs and the constant devising of new exploratory methods; the unaccountable losses in smelting and the continual change in treatment-practice. For these anomalies certain climatic conditions seem to be chiefly responsible.

In desert regions, and in those cases in which vadose ore-bodies obviously have no direct association with true fissure-veins, the metallic content appears to occur mainly in the interstitial clays of conglomerate layers, brecciated and sheared belts, joint- and fracture-crevices, and fault-planes. Probably four-fifths of the mining-prospects are founded on such conditions, and many paying mines are opened upon such indications, which not unfrequently have led to bonanzas. It is, however, the great bodies of disseminated ore that will command hereafter the greatest attention. When permanent water-level is reached, the values shown in such outcrops usually soon disappear, or else the mineralogic character of the ore abruptly changes, necessitating more or less complete alteration or even complete replacement of both mining-method and mill-treatment.

For example, it has long been the custom, in treating gold-ores of this class, to crush all of the "vein-rock," and then separate the values. That the values actually are located in the interstitial clays is shown not only by careful panning, chemical analysis, or microscopical examination, but by the results of the dry separators, especially the recently-invented Quenner pattern,[1] the construction of which is based upon the trommel-screen, and which has been so successfully worked on the cement-conglomerates of the Altar gold-mining district near El Tiro, in Sonora, Mexico.

I do not know of any attempt to explain in detail the features of these deposits. Miners regard the full width of the metal-bearing belt as the "vein." Sometimes, as in the cases of distinct recognizable fault-planes, the clay selvage is shown to be the streak richest in values. In the descriptions of mines there occasionally appears a hint that this phenomenon has incidentally attracted notice. My personal opinion is that the principle involved in the localization of values in the clay-seams has a definite rôle and a wide field in ore-genesis generally; and that, in dry regions at least, the localization of the values is due partly to the prevalency of chloride compounds and partly to the easy reduction of such compounds in contact with the alkaline silicates composing the clay gouges.

[1] *Engineering and Mining Journal*, vol. lxxxix., No. 17, p. 858 (Apr. 23, 1910).

In many cases, in which the values are high and the clay bodies are sufficiently large to form the principal ore-mass extracted, the smelter-returns prove disappointing. Careful chemical analysis gives results higher in the precious metals than the ordinary assay-figures. In the instances mentioned elsewhere, discrepancies of this kind finally led to an examination of the assay-methods in vogue, the results of which clearly indicate that these methods are often faulty when high-grade ores are involved.

II. Gossan-Zone in Dry Climates.

1. *Peculiarities of Arid Gossans.*—Under conditions of aridity, gossan-formation and rock-decay present many novel contrasts to the phenomena of this class displayed in normal moist districts. Particularly striking are: the great depths to which the gossan-zone extends; its apparently inexplicable irregularity in thickness; the richness of the superficial ores, or their aggregation in bonanzas, which adds excitement to their exploration; and the varying and unusual mineralogic character of the ores, which complicates their metallurgic treatment. There are many other interesting features of minor importance.

The formation of gossan, being merely a special phase of general rock-decay, is directly and profoundly influenced by climate. As Russell[2] long ago pointed out, rock-decay in general appears to be the direct result of normal climatic conditions; in cold or arid regions the rocks are scarcely at all decayed. This climatic control of general erosion has a special effect upon gossans. One noteworthy feature is that while in a moist climate rock-decay almost everywhere goes on faster than the decomposed materials are removed, under conditions of aridity the reverse is true. In the latter case, the breaking-down of rock-masses is mechanical rather than chemical, through a process technically termed insolation, and, as I have elsewhere shown,[3] the finer rock-waste is at once carried off by the winds almost as fast as it is formed.

Paradoxical as it may seem, arid regions exhibit a practical absence of general rock-decay, but gossans of exceptionally great depth. Mineral veins, fault-planes, and sheared belts

[2] *Bulletin of the Geological Society of America*, vol. i., p. 134 (1889).

[3] *Bulletin of the Geological Society of America*, vol. xix., p. 63 (1907).

appear to be the only lines along which chemical decay of the rocks of desert tracts is in the least appreciable.

Wind-scour, or deflative action, much more than water-action, tends to leave the heavy minerals behind. A soil especially rich in ore-materials results. To this are added the constant contributions from space.[4] Metallic minerals, instead of being converted at once into soluble form and carried away by surface-waters, must find their way largely into gossans. In solution, they then percolate into the pores, crevices, and other cavities in the country-rock, often enabling the latter to be profitably worked as ore. The gold-veinlets of the Ortiz laccolith, in central New Mexico, as described by Yung and McCaffery,[5] are not exceptional examples. I have also recently ventured to suggest[6] some of the reasons for the so-called porphyry-coppers being necessarily so characteristic of arid regions. This phenomenon is actually one of wide extent.

2. *Remarkable Depth of Desert Vadose Zone.*—As previously stated, in excessively dry climates the gossan-formation often presents side by side its two extreme facies. It is no uncommon occurrence that unaltered sulphides are exposed to the sky, while the superficial alteration of near-by ore-veins has proceeded to depths sometimes exceeding 1,000 ft. In the majority of cases, the depth of the gossan is to be measured by hundreds of feet. Upon a radical revision of the usual interpretation of these conditions seems to rest the chief hope of the thousands upon thousands of shallow mining-prospects scattered throughout the arid regions, from British Columbia to Patagonia, and the explanation of the bonanza-deposits so famous, for instance, in Mexico and Peru, where, until recently, on account of the difficulties encountered in handling the water of the deeper mines, it has been impossible to extract ores from depths much below the ground-water level.

Whether the sulphide zone lies 200 or 300 ft. beneath the surface, as in many parts of Montana and Nevada, or 500 or 600 ft., as in many places in New Mexico and Arizona, or 1,000 ft., as in central Old Mexico, or 1,500 ft., as in Chile, it is

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[4] *Trans.*, xli., 153 (1911).

[5] *Trans.*, xxxiii., 358 (1903).

[6] *Bulletin of the Mining and Metallurgical Society of America*, vol. ii., No. 25, p. 316 (July, 1910).

possible to recognize instructive relationships between the depth to which ore-veins and shattered belts undergo alteration and the general surface of the country-rock affected by chemical decay.

It seems probable that in desert regions, or wherever the annual rainfall is less than 10 in., relatively little meteoric water penetrates to the depths, except through the larger fault-planes and sheared belts. That old mineral veins, through which there is no longer flowage of surface-waters, apparently do not decay faster than the surrounding country rock, is a frequent observation, explaining the unexpected occurrence at the surface of sulphide ores with no signs of alteration. For example, veins of galena in the Sierra de los Caballos, on the Rio Grande, display mineral as fresh at the surface as it is 1,000 ft. below, at ground-water level. The same is true of the Oro Blanco district in southern Arizona, and the Spring Mountain in central Nevada; and these are by no means isolated instances. Zinc, copper, and silver sulphides exhibit the same phenomenon.

On the other hand, the influence of recent faulting upon the extent of gossan-formation is well shown in those veins which are located along lines of differential movement, and where it is perhaps still in progress. Even more strikingly is the phenomenon displayed in those cases in which an old vein is crossed obliquely by recent fault-planes. When sufficiently far apart, such lines of displacement give rise to the apparent anomaly of two or more rich gossan-zones one above another, and separated by considerable zones of unaltered sulphide vein-stuff. Through differential movement in a vein, or even in the inclosing rock-mass, the gouge or selvage produced appears to have a prime influence in the localization of ore-materials, or the enrichment of the vein already present. Such enrichment is entirely within the zone of gossan-products.

3. *Bonanza Ore-Blanket of Arid Regions.*—In the normal fissure-vein three distinct ore-zones are usually recognizable: (1) a relatively thin zone of oxidized ores of the gossan at the top; (2) a sulphide-enrichment zone, commonly only a few feet in thickness, in the middle; and (3) at the bottom a lean unaltered sulphide-zone extending to indeterminable depth. In arid regions the first and second zones are greatly expanded;

so that several other subordinate zones are easily distinguishable. Fuchs and DeLaunay,[7] for example, note no less than six well-defined zones in the Mexican silver-regions. Under the conditions of a moist climate, the gossan is so limited in depth that these several ore-zones are commingled; even the bonanza-zone at ground-water level is often barely distinguishable.

Viewed broadly, it seems likely that the vadose zone will have to be regarded much in the same way as we now regard the regolith—a bonanza zone being everywhere present at the bottom, in some places well developed, but in others only feebly indicated by ore-materials. The zone of secondary sulphide enrichment thus appears as a universal and not merely a local phenomenon.

According to the prevalent notion of ore-formation, the secondary enrichments of mineral veins so often found at permanent water-level are regarded as dependent for their metallic materials mainly upon leaching from the upper weathered portions of the vein itself. This is, however, merely a special phase of a more general process. The metallic leachings may come, not from the superior part of the vein at all, but from veins, veinlets, and the country-rock itself, for a considerable distance around. Nor need the leachings come directly down the course of the vein; they may travel obliquely along fault-planes or joint-planes; they may be transported laterally along stratification-planes or porous layers; or, without any direct connection with other ore-bodies, they may percolate along shearing-planes and through shattered belts, or even be impounded by dikes and faults. Recent observations suggest the advisability of a complete reinvestigation of the formation and localization of ore-bonanzas.

4. *Variety of Dry Gossan-Ores.*—Since in moist countries the gossan-zone is usually relatively thin, it is exceedingly difficult, and at times even impossible, to observe its separation into distinct stages of ore-transformation. Only occasional glimpses are caught of the processes involved. Of the numerous and complex chemical reactions known to take place, merely the first and the last are noted. All of the intervening changes are inextricably intermingled, and at best can only be surmised. There is little intimation of systematic sequence.

[7] *Traité des Gîtes Minéraux et Métallifères*, vol. ii., p. 816 (1893).

In arid lands the gossan-zone is so thick that the several successive stages of ore-alteration are readily made out. Unusual minerals occur in such abundance as to form of themselves notable ore-bodies. At Magdalena, in Socorro county, N. M., such rare minerals as chalcophanite, aurichalcite, chalcanthite, and hydrozincite reach exceptional development,[8] and the zinc-replacement ores are even more remarkable.[9] The complexity of both the oxidation-products and other ore-materials is indicated in many ways. Its broader features are recognized by the Spanish-American miners in their zonal arrangement of *colorados*, *mulatos*, and *negrillos* ores.

The more scientific attempt at zonal differentiation by Fuchs and DeLaunay[10] has been already referred to. These authors find the following mineralogic succession in the Mexican silver-mines, the subdivisions being referable to the three main zones above recognized:

Gossan-Zone.	1. Native silver, in iron oxide and manganese oxide; moderately rich. 2. Chloride, bromide, and iodide of silver, with some native silver, and iron and manganese oxides; moderately rich.
Bonanza-Zone.	3. Silver sulphide, with some black antimony sulphide ore; very rich.
Primary Zone	4. Black silver-antimony sulphide (pyrargyrite, and proustite). 5. Gray copper, blende, etc. 6. Blende, pyrite, quartz, in mixture.

Theoretically, the changes taking place during the oxidation of metallic sulphides should be controlled largely by the chemical affinities of the bases for oxygen. Moreover, the minerals formed at any particular stage should be dependent not only on the nature of the bases originally present, but on the character and number of bases subsequently introduced.

Primary ores which are complex aggregates of the sulphides of iron, copper, lead, zinc, and silver, and which so abound throughout arid America, owe their great variety of gossan-ores to climate. Under climatic conditions so abnormal, the metallic salts in solution are in a relatively high state of concentration, and the oxidation of primary ores does not take

[8] *Lead and Zinc News*, vol. ix., p. 6 (1904).

[9] *Mining Magazine*, vol. xii., No. 2, p. 109 (Aug., 1905).

[10] *Traité des Gîtes Minéraux et Métallifères*, vol. ii., p. 816 (1893).

the usual course. In moist climates the decomposition of metallic sulphides ordinarily results in the formation of sulphates, which are then rapidly changed into more stable oxides and carbonates, while chloridic compounds do not appear to play an important part. Thus, in the same mass of complex primary ore, subject to the same vadose waters, the alteration-products remaining may be hydrous iron sesquioxide, basic copper carbonate, lead sulphate, zinc silicate, and silver chloride. But under very dry conditions, copper sulphate, ordinarily too soluble to assume the crystal state, becomes an ore; zinc carbonate is dominant, and silver chloride is often a principal ore. Iron chloride and copper chloride may also be found with the silver chloride, but the former is still so easily soluble that it is soon removed by the scanty circulating waters, while the latter often assumes solid form in the mineral atacamite.

III. Prevalency of Chloridic Ores in Arid Regions.

1. *Distinctive Stages of Haloid Ore-Formation.*—One of the most remarkable features concerning the gossans of excessively dry regions is the abundance of certain chloridic ores, of which horn-silver is perhaps the most familiar. Other minerals of the same class actually occur more frequently than is generally supposed; but there are many good reasons for their not being so well known. In the main, metallic compounds of the chloride group are more or less easily soluble. Under ordinary moist-climate conditions, they hardly occur at all as gossan-ores, but representing a relatively transitory stage, soon pass into some other more permanent state. Only in the desert country do they constitute definite ore-materials. There, however, they furnish the key to the solution of many of the great general problems of ore-deposition.

Recent observations on the origin of the horn-silvers in dry regions throw light also upon the formation of the less familiar ores of the same class. The abundance of cerargyritic ores in certain regions is now regarded as due mainly to unusual climatic influences.[11] The great commercial importance of these haloid silver-ores, so strongly emphasized in the history of mining, is not wholly explained by the fact that horn-silver is merely one of the less soluble of the chloridic compounds.

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[11] *Trans.*, xxxix., 166 (1909).

It is also one of the most characteristic ores of arid districts. That it should be so prevalent in desert regions is not so strange after all, since chloridic material, in the form of common salt, is everywhere furnished by the wind-blown dusts[12] off the salt-flats and from the saline soils of the intermontane plains. In quantity for a given area, the saline matter thus furnished through æolic means very greatly exceeds that which is supplied, under normal conditions, through the decomposition of the rocks. Saline dusts deposited on the surface are acted upon by the first rain; and large amounts of salt are thus carried down to mingle with the underground waters. This introduction into the gossan produces conditions exceptionally favorable to the formation of haloid compounds of the metals. There is nothing at all comparable to this agency in moist countries.

2. *Occurrence of Chloridic Ores.*—Of all the haloids of the metals found in nature, chlorides alone are familiar to us as ores. Yet, notwithstanding their importance, the manner of origin of the silver chlorides has remained one of the very last problems to be understood. As recently noted,[13] none of the older explanations adequately account for the great abundance of this class of ores throughout the arid regions, and for their conspicuous absence elsewhere. All things considered, the most satisfactory hypothesis is that the chlorine required for their formation has been amply supplied from the saline soils which are constantly blown about in large quantity in all desert tracts.

In considering the formation of the horn-silvers, it seems wholly unnecessary to rely upon the supposed action of sea-water upon vein-outcrops, as urged by Mösta[14] for certain Chilean ores, or by Henwood[15] for the veins of the Little Shack mines on Manche, off the coast of Normandy; or the direct action of sea-water upon such materials as the slag-heaps of Laurium, in Greece, as suggested by Brauns;[16] or the action of saline lake-waters, as proposed by Ochsenius,[17] or the similar

[12] *Economic Geology*, vol. ii., No 8, p. 778 (Dec., 1907).

[13] *Trans.*, xxxix., 163 (1909).

[14] *Vorkommen der Chlor-, Brom-, und Jod-Verbindungen des Silbers* (1870).

[15] *Metalliferous Deposits*, vol. i., p. 530.

[16] *Chemische Mineralogie*, p. 367 (1896).

[17] *Die Bildungen der Natronsalpeters aus Mutterlaugensalsen*, p. 51 (Stuttgart 1887).

theory of Sandberger[18] for the Peruvian deposits of the Cerro de Huantajaya; or that of Penrose[19] for certain deposits of the arid parts of the United States.

Little as the other metallic chlorides have been regarded in mining-operations, their frequency and abundance are often somewhat surprising. In the friable gossan-ores of Lake Valley, N. M., blackened by iron and manganese, the chloro-bromide of silver is almost as plentiful as horn-silver.[20] At the Torrance mine, near Socorro, there are associated with the gray cerargyritic ore the grayish-green chloro-bromide of silver (embolite) and the pale yellow iodide of silver (iodyrite). The relatively large amounts of the bromide and iodide compounds in such occurrences may be sometimes due, as Kosmann[21] has suggested, to their greater insolubility as compared with the chlorides; but under dry-climate conditions this does not seem to obtain.

Since the saline dusts in desert regions appear to be so intimately associated with the formation of haloid silver compounds, it might be fancied that they would be also highly influential in the chemical transformations of the chemically-kindred metals, gold and copper. Exact investigation in this direction is a wholly new field. As yet, little has been done to even suggest the lines of most fruitful inquiry. The unusual prevalency of gold in arid gossans would indicate that the conditions must be quite as favorable for the precipitation of this metal as for silver, and that saline dusts play an important rôle as an immediate source of the chlorine.

In spite of the fact that the copper chloride is seldom listed among ore-minerals, it is somewhat common in arid districts. In moist climates it is, of course, almost unknown except in solution in some mine-waters. It doubtless performs a far more important function in ore-genesis than is generally conceived. In this rôle it appears to be second only to the sulphate. For several reasons, it is apt to escape notice. Cupric chloride, $CuCl_2$, is quite soluble in water, and only in very dry places appears at all as a crystallized mineral. Cuprous chloride,

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[18] *Neues Jahrbuch für Mineralogie, Geologie, und Palaeontologie*, p. 174 (1874).

[19] *Journal of Geology*, vol. ii., No. 3, p. 314 (Apr.–May, 1894).

[20] *Trans.*, xxxix., 159 (1909).

[21] *Leopoldina*, vol. xxx., p. 1 (1894).

Cu_2Cl_2, although rather insoluble, is, on account of its white or gray color and earthy texture, not easily detected among clays and other gossan-matter.

Atacamite, the copper oxychloride, $Cu_2Cl(OH)_3$, is also widely distributed in arid regions; but, on account of its green coloration, is generally mistaken for malachite. It frequently constitutes a considerable proportion of the green gossan-ores. In Chile and Peru this mineral, according to Murdoch,[22] forms important ore-bodies. Rickard[23] reports it as occurring throughout the southwestern United States. Long ago, Field[24] and Friedel[25] produced this mineral artificially by methods not very unlike those of nature.

That atacamite is not by any means an uncommon mineral is shown by the circumstance that other copper-minerals are found as pseudomorphs of it. Bärwald[26] has even reported pseudomorphs of the silicate of copper, chrysocolla, after atacamite, and Tschermak[27] has described malachite derived from atacamite. These citations are sufficient to draw attention to the point that in excessively dry regions the chloridic compounds of copper, in all likelihood, perform a function not unlike that of the similar silver salt, and that the chloridic materials in desert-dusts are ample to serve the purpose of supplying the necessary chlorine.

The far-reaching influence of the chlorides of the metals in ore-formation has been overlooked, mainly because of a general proneness at the present time to ascribe to the easily-soluble sulphate combinations the chief importance in the transitory stages, and partly because the chemistry of ores has been largely investigated under conditions of moist climate, where the metallic chlorides are seldom encountered.

In the process of ore-formation during the cooling of magmas, it appears, as indicated by Clarke,[28] that chlorides perform

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[22] *Transactions of the Institution of Mining and Metallurgy*, vol. ix., p. 300 (1900–01).

[23] *Trans.*, xxxi., 206 (1901).

[24] *Philosophical Magazine*, Fourth Series, vol. xxiv., No. 159, p. 123 (Aug., 1862).

[25] *Comptes rendus*, vol. lxxvii., p. 211 (1873).

[26] *Zeitschrift der Krystallographie und Mineralogie*, vol. vii., p. 169 (1883).

[27] *Jahrbuch der kaiserlich-königlichen geologischen Reichsanstalt*, vol. xxiii., No. 1, Mineralogische Mittheilungen, p. 41 (1873).

[28] *Bulletin* No. 330, U. S. Geological Survey, p. 549 (1908).

definite functions. When, as temporary carriers of the metals, their work is done, they enter into other combinations. In the vadose reactions, especially under arid climatic conditions, metallic chlorides seem to serve an analogous function, and in this respect to take largely the place of the sulphates.

Another reason for the relatively small attention given to the metallic chlorides is, doubtless, the fact that it has been hard to imagine exactly how, from common salt, chlorine could be readily liberated, so as to react upon most of the metal-bearing minerals, particularly those containing gold and copper. This phase of the problem has been partly solved experimentally by Don,[29] who has shown that, in the presence of manganese oxide, chlorine is readily set free from hydrochloric acid, the latter being formed through the decomposition of pyrite. The far-reaching significance of this explanation applies particularly to arid regions, where manganese is unusually abundant. The blackened and burnt aspect of desert rocks, so well described by Wallace,[30] is ascribable largely to a superficial deposit of iron and manganese. Blake,[31] in particular, notes this phenomenon in Arizona and California. The black, earthy, and often pulverulent, gossan of many arid ore-deposits is commonly impure manganese oxide. In the horn-silver deposits[32] of Lake Valley, N. M., the real character of the ore is disguised by this black material. The same associations are frequently met with throughout the dry regions. In connection with the gold-veins of Colorado, Rickard[33] also refers to the significance of the presence of earthy psilomelane in considerable amount. In arid regions, at least, this mineral doubtless plays a hitherto unexpected, yet leading, part in the formation of vadose ores generally.

3. *Peculiarities of Haloid Ores.*—Some of the most striking features of the horn-silvers, as displayed under climatic conditions of aridity, I have recently described at length.[34] The reactions of copper-salts under like conditions deserve more attention than has ever been given them or than can be devoted to them here. That the chlorides perform an important function in the formation of the disseminated ores of copper cannot

[29] *Trans.*, xxvii., 599 (1897).
[30] *Land of the Pueblos*, p. 140 (1888).
[31] *Trans.*, xxxv., 371 (1905).
[32] *Trans.*, xxxix., 159 (1909).
[33] *Trans.*, xxxi., 207 (1901).
[34] *Trans.*, xxxix., 159 (1908).

now be well doubted. As was pointed out a short time ago,[35] the disseminated-copper deposits appear to be strictly characteristic of arid regions; and the localization of ores of this class is thus singularly dependent upon climate. In many ways, the physical conditions presented by deserts are exceptionally favorable to the formation of extensive ore-blankets of disseminated character. Copper-deposits, however, need not be considered at length here. The relations of gold to chloride compounds are discussed in another place.

One of the prime factors in the formation of the haloid deposits seems to be the chloridic state of the ore-materials; and another factor appears to be the reducing action of the silicates in interstitial or gouge-clays. To these features special attention is here directed.

Other points connected with the haloid ores which have been more fully discussed elsewhere,[36] are: (1) that often they have no direct associations with either eruptive masses or distinct fissure-veins; (2) that the ore-bodies are more directly dependent upon geologic structures than is commonly the case; (3) that their deposition is frequently determined by peculiarities of the local surface-relief; (4) that the relatively high concentration of mine-waters of arid regions must profoundly affect the precipitation of their contents; and (5) that the usual explanations of the origin of the haloid ores have but limited application in dry countries.

IV. Origin of Certain Gouge-Ores.

1. *Formation of Gouge-Materials.*—The kaolinization of feldspathic rocks probably has a greater influence upon the localization of ore-deposits than has been surmised hitherto. The presence of kaolin as *a priori* evidence of downward sulphide-enrichment has been recently emphasized by Lindgren,[37] Emmons,[38] and others. The recognition of this association is of far-reaching importance, and in its bearing upon vadose ore-genesis is of much greater significance than any of these authors have been inclined to ascribe to it.

[35] *Bulletin of the Mining and Metallurgical Society of America*, vol. ii., No. 25, p. 316 (July, 1910).

[36] *Trans.*, xxxix., 139 (1909).

[37] *Economic Geology*, vol. ii., No. 2, p. 120 (Mar.–Apr., 1907).

[38] *Idem*, vol. v., No. 5, p. 477 (July-Aug., 1910).

Although, as commonly applied, the terms fluccan, selvage, and gouge, refer to the thin layer of unctuous clay which often lies between the vein-matter and the wall-rock, the clayey materials found in joint-cracks and in brecciated belts bordering lines of dislocation are genetically the same; so that the name gouge-material may be appropriately made to cover all of the clays thus associated. These clays are, of course, produced chiefly through the slow chemical decay of the separated rock-faces; and, in the case of most eruptives at least, the process is mainly a kaolinization of the feldspars.

In most regions, on account of the relatively thin gossan and the great abundance of partly-decomposed rock-débris scattered through it, kaolin gouge-materials and clay selvages in rock-crevices and fault-planes are inconspicuous, and rarely receive particular notice in mining-operations. In arid country, where the gossan-zone assumes great depth, and general rock-decay is almost unknown (except along fault- and joint-planes and in brecciated belts), the soft interstitial gouge-clays are noteworthy features, and almost invariably carry mineral values, thereby attracting the special attention of miners. In other respects also, this fact is significant.

Along fault-planes, thick selvages are formed partly by rock-flour produced by the grinding movement of wall-faces, and partly by the introduction of kaolinized material from above. Whether in fault-plane, joint-crack, or brecciated belt, the clay selvage forms a plate, as it were, more or less impervious to ground-water.

2. *Values in Gouge-Bands.*—Although gouge-clays have often been disregarded or discarded as worthless in mining-operations, they may sometimes carry not only high values, but the only values in the "ore" mined. When closely associated with well-defined fissure-veins, the gouge-matter and its values are very apt to be neglected or overlooked. This is especially true when, as is not infrequently the case, the vein itself has undergone repeated movement parallel to its walls.

In many brecciated belts in eruptive rock-masses the barren rock-fragments included in the so-called "ore-bodies" might well be separated from the interstitial clay, thereby concentrating all of the values in the muds washed away into the settling-pools. Among the Oro Blanco deposits in southern

Arizona a brecciated ore-body was opened, the average gold-value of which, for its whole width of 20 ft., was about $8 per ton. When the values were carefully limited, 16 ft. of this width was found to carry less than $3 per ton, while the 4 ft. next the selvage yielded $15, and the gouge, less than an inch in thickness, $180 per ton. Similar conditions obtain in the Ortiz mountains, in central New Mexico, although there the country-rock, which is mainly monzonite, as indicated by the recent chemical analyses of Ogilvie,[39] itself assays about $1 per ton in gold.[40]

Some of the so-called "tallow-clays" of the southwestern Missouri lead- and zinc-fields appear in a similar rôle. These "clays," in their present condition, are essentially aluminum and zinc silicates, carrying, according to Seamon,[41] often as much as 54.92 per cent. of zinc oxide. Attention was early called by Chauvenet[42] to the zinciferous character of these "tallow-clays," which fill joint-planes and stratification-planes and sometimes broaden out into deposits several feet thick.

The occurrence of copper in gouge-clays is more frequent than has been generally supposed. Black copper especially, as many assays have shown, is found in clays even when its presence is least expected. In the same way, the clays of certain of the Ely, Nev., copper-deposits contain chalcopyrite.

The silver-content of gouge-clays is so notorious that they are quite generally tested for this purpose. In arid districts, as at Kingston, Hermosa, Chloride, and elsewhere in the Mimbres range along the continental divide in New Mexico, and in many of the silver-camps of Old Mexico, the gouge-clays carry very high values. Some of these instances are considered in the assay-notes appended. Occurrences of this kind are so numerous, and the metallic constituents are so varied, as to suggest a closer genetic relationship between the clayey materials and their contained values than is involved in the mere percolation of meteoric waters down the cracks in the country-rock.

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[39] *Journal of Geology*, vol. xvi., No. 3, p. 230 (Apr.–May, 1908).

[40] *Trans.*, xli., 148 (1911).

[41] *American Journal of Science*, Third Series, vol. xxxix., No. 229, p. 40 (Jan., 1890).

[42] *Missouri Geological Survey, Report of* 1873–4, p. 409 (1874).

In this connection, two phenomena demand particular consideration. One is the dialytic action exhibited by soils and clays, and the other is the reduction by alkaline silicates of the metallic salts dissolved in ground-waters. Both of these chemical activities may have special functions in ore-genesis, but the latter is believed to be the more potent, particularly under arid climatic conditions.

3. *Dialytic Rôle of Selvages.*—The accumulation of metallic salts in the thin layers of somewhat porous gouge-clays has been explained on the basis of the phenomenon known as adsorption. Through such porous layers water rather freely passes; but many substances dissolved therein do not. This action is shown by a familiar experiment in the chemical laboratory.

Separating by a porous layer of asbestos solutions of silver nitrate and aqueous hydrochloric acid, and allowing them slowly to commingle, Kuhlmann,[43] was able long ago to produce a silver chloride having the appearance of horn-silver or cerargyrite. More recently, Clarke[44] has expressed the opinion that such a blending of solutions may also take place in nature through layers of decomposed rock substances, such as sandy clay or gossan. The familiar laboratory-phenomenon of the dialytic separation of solutions was also incidentally suggested in 1893 by Becker,[45] as probably explaining the genesis of certain of the quicksilver-deposits of California. More specific in their bearing upon ore-deposition are the extensive experiments of Kohler,[46] who filtered various salt-solutions through kaolin. Briefly stated, the theory is that there is a selective concentration, as it were, of the minerals in solution on the surface of the solid. More recent investigations do not fully support this suggestion. As shown by Rohland,[47] the filtration of solutions through clays presents very different results, according to whether the solute is a colloid or a crystalloid. In the first case the solute does not pass through; in the second, it does.

In an inquiry into this subject undertaken several years ago in the chemical laboratories of the New Mexico School of

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[43] *Comptes rendus*, vol. xlii., p. 374 (1856).

[44] *Bulletin* No. 330, *U. S. Geological Survey*, p. 564 (1908).

[45] *Mineral Resources for* 1892, *U. S. Geological Survey*, p. 156 (1893).

[46] *Zeitschrift für praktische Geologie*, vol. xi., p. 49 (Feb., 1903).

[47] *Zeitschrift für Elektrochemie*, vol. xi., No. 28, p. 455 (July 14, 1905).

Mines, I proceeded upon the hypothesis of adsorption. The clay materials used comprised natural gouges (some of which were already highly metalliferous), and also clays without notable metallic content. It was soon found that, in nearly all cases, marked chemical reactions took place, and that it was very doubtful whether actual adsorption entered into the problem at all. At that time, the extensive experimentations of agricultural chemists along similar lines had not come to my notice.

These results tend to negative the notion that dialysis directly promotes ore-deposition. But the conditions imposed in nature may be such as to modify this conclusion. Selvages may serve as local or temporary retardants of ground-water currents, thus producing impoundment, greatly favoring ore-deposition. This consideration is especially probable in excessively dry regions, where the metallic salts are held in much more concentrated solution than where there is always an abundance of moisture in the vadose zone, and for this reason, many chemical reactions, unknown elsewhere, may take place.

4. *Accumulation of Metallic Salts in Gouge-Clays.*—The interchange of chemical elements during the process of general rock-alteration was early recognized by geologists. It is, however, to the agricultural chemists that we owe the key to the explanation of the gouge-ores. Both the guiding principles and the immediate aims of their work apply strictly to the vadose conditions of ore-formation. When meteoric waters carrying metals in solution come in contact with decayed rock-faces, or with the kaolinic products of rock-decay, there is a notably selective retention of elements.

More than half a century ago the English chemist Thompson [48] found that ordinary waters, in passing through soils and clays, took from the latter some substances and left others. It was further pointed out by Way [49] and Eichhorn [50] that when salt-solutions were brought into contact with certain soils and kaolins there was an interchange of bases. Later, since Lemberg [51] went so exhaustively into the subject of this interchange

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48 *Journal of the Royal Agricultural Society*, England, vol. xi., p. 68 (1850).

49 *Journal of the Royal Agricultural Society*, England, vol. xi., p. 313 (1850).

50 *Poggendorff's Annalen der Physik und Chemie*, vol cv., p. 126 (1858).

51 *Zeitschrift der deutchen geologischen Gesellschaft*, vol. xxviii., p. 519 (1876).

of bases, and Dittrich [52] and Van Bemmelen [53] continued similar lines of investigation with so many interesting results, a number of agricultural chemists have devoted their attention to the same subject; so that, so far as soils and clays are concerned, the basic principles involved are now well established.

5. *Precipitation of Ore-Materials by Silicate Minerals.*—As distinguished from the strict hypothesis of adsorption, the possibility of an extensive reduction of metallic salts in solution, when brought into contact with finely-divided silicates in clays, has much to support it. The investigations conducted along these lines at the New Mexico School of Mines have been already mentioned. Sullivan,[54] taking his cue from the agricultural chemists in their experiments with soils, has recently obtained some suggestive results concerning the interaction between powdered minerals and water solutions, with special reference to its bearing upon ore-formation. He used finely-divided kaolin, shale, feldspar, pyrites, and biotite, and a solution of copper sulphate. His most noteworthy observation was the great extent to which the powdered silicate-minerals removed the copper from solution. The reactions were demonstrated to be chiefly an exchange of bases; copper being precipitated, and an equivalent quantity of other bases (mainly the alkalies and alkaline earths) entering the solution. In a later statement [55] the same author observes that the fact of prime significance geologically seems to be that, by a process of simple chemical exchange, the metal may be removed from solution and fixed in the solid state, and thus concentrated, by contact with even the most stable of the silicates. The bases most commonly replacing the metals in these processes are potassium, sodium, magnesium, and calcium.

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[52] *Mittheilüngen d. Gr. Badisch geol. Landesanstalt,* vol. iv., p. 339 (1903).
[53] *Zeitschrift für anorganische Chemie,* vol. xxiii., No. 4, p. 321 (May 4, 1900).
[54] *Economic Geology,* vol. i., No. 1, p. 67 (Oct.–Nov., 1905).
[55] *Bulletin* No. 312, *U. S. Geological Survey,* p. 61 (1907).

Assay of Silver-Bearing Gouge-Ores.

BY CHARLES R. KEYES, DES MOINES, IOWA, AND D. F. RIDDELL, PARRAL, MEXICO.

(Wilkes-Barre Meeting, June, 1911.)

I. Introduction.

For a period of several years, and in a large number of cases, the Metallurgical Laboratories of the New Mexico School of Mines were employed in umpire work. During this time many important local problems were solved. Against the results of careful chemical analyses of ores, assay-returns were often checked. In many cases the assay-methods of mine and custom-smelter were compared, and both often found faulty for the ore concerned. A wide range of ores was covered. Many of the chemical analyses and assays were made by Dr. F. C. Lincoln, Professor of Metallurgy, now of the Montana State School of Mines. A varied series of instructive assays was prepared by Prof. R. B. Brinsmade, now of the West Virginia State University. Single assays and given series of tests were undertaken by H. T. Goodjohn, now chief chemist to the Cia. Metalurgica de Torreon, Coahuila, Mexico; by H. J. Hubbard, now superintendent of the Butters Divisadero Mines, Jocoro, San Salvador, C. A.; by E. D. Morton, now superintendent of the Arizona & Nevada Copper Mining Co., at Lunning, Nev.; and by A. W. Edelen, now superintendent of the Angangueo Unit of the American Smelting & Refining Co., in Michoacan, Mexico. A critical comparison of assay-methods that were found to be followed in the case of certain gouge-ores mined in the Mimbres range in New Mexico, and about which there had been much controversy, was made by D. F. Riddell, then Acting Professor of Assaying, and now superintendent of the Providentia Mines Co. of Parral, in Chihuahua, Mexico. To him is due the credit of much that is contained in the following notes, which he has generously permitted to be used in advance of the publication of the complete results.

The Mimbres ore is representative of a large number of somewhat complex ores, high in silver, from the dry region of the southwestern United States and northern Mexico. Among other uncertainties connected with many of these ores, assay-determinations checked poorly; the values in precious metals, as returned by the smelters, were frequently too low, and on account of their variant character, even in the case of the same ore, the assay-results were very unsatisfactory.

Among smelters generally, the prevailing opinion is that in a slag which is either extremely acidic or extremely basic in character there is a greater or less loss in silver-values. Some assayers have the same notion. Among them also there is the belief that the use of borax in assaying causes low results for the silver-content. On the other hand, the majority of assayers appear to regard the use of borax as removing at once all of the difficulties which in any way arise from imperfect fluxing in the crucible. From experience, the inclination is to regard many charges, and slags, which by the assayer are ordinarily called good, especially when they contain much borax, *i. e.*, 0.5 A-T., or more, as not, in case of moderately high-grade ores, checking well with one another. The usual methods of assay of ores running high in copper and sulphur are also very unsatisfactory.

The series of determinations, of which the single silver-ore here considered in detail is a type, had for its immediate objects the experimental proving of the effects upon the silver-yield of an excess of each of the fluxes ordinarily used in the crucible of the assay-laboratory, and of showing exactly what type of charge gives the best results on an ore which is high in copper and sulphur, and which at the same time is a typical roasting-ore.

In the ore under consideration the silver-content was already known to be high. By experimentation it was determined, (1), how acidic a charge could be run; (2), how basic a charge could be run; (3), how large an amount of borax could be used in a charge; (4), how the normal charge for the kind of ore would behave; and (5), how the use of an excess of litharge would affect the results.

This ore was much more difficult to handle than many of those found in the region. The large excess of niter, KNO_3,

which it was necessary to use was checked against the dead-roasted ore. In each series the attempt was made to get five results that would check within 1 in 100; and to take the mean of these best checks as the normal for the charge used. A concentration-test was made on the ore, and the reducing-power and value of the concentrates determined in order to find whether or not the value, the sulphur, the weight of the ore, and the weight of the concentrates, had any definite relations to one another. It is to be noted that whenever the word "borax" is mentioned, the variety known as "borax glass" is referred to, and all crucibles received a cover of 10 g. of borax before fusion; that all buttons of the same ore were worked as nearly as possible under the same conditions, and all were scorified with the same weight of test-lead; and that all buttons were cupelled so as to obtain "feathers."

II. Preparation of Samples, and Chemical Analyses.

In preparing the ore 32 lb. was pulped and put through a 100-mesh sieve, and then mixed by rolling on oil-cloth and sifting through a coarse sieve. The separated scales gave a bead of 83.4 mg. of silver and a trace of gold. In all of the assays this ore gave only a trace of gold. By panning 100 g. of ore, 44.9 g. of concentrates, mainly sulphides, were obtained.

A part chemical analysis of the ore-sample ready for slagging gave:

	Per Cent.
Iron,	8.7
Lead,	tr.
Copper,	25.6
Zinc,	6.2
Manganese,	1.0
Lime,	15.0
Sulphur,	10.0

III. Reducing-Power of the Ore.

In determining the reducing-power of the ore, the charge, with borax cover, was as follows: Ore, 2.00 g.; PbO, 1 A-T.; $NaHCO_3$, 0.75 A-T. The lead-button obtained weighed 7.105 g., the reducing-power of which was 3.6, the oxidizing-power of the niter being 3; therefore $\{[(7.105 \div 2)\ 14.5] - 15.0\} \div 3.0 = 12.0$ g. of niter per charge. This charge of niter, KNO_3, is excessive, but there appears to be no other way

of handling this ore easily. If properly run, the excess of niter is not so deleterious in its effects as those of roasting, which is the only other practical method.

IV. Assay of Dead-Roasted Ore.

In the case of the dead-roasted ore the charge, with borax cover, was: Ore, 0.5; PbO, 1.5; $NaHCO_3$, 0.75; SiO_2, 0.2 A-T.; and argols 2 g. There was secured a good slag which poured cleanly. The buttons were hard and coppery; these were scorified with 20 g. of test-lead. The cupels displayed a coppery tint. The first bead weighed 103.06 mg.; the second bead, 103.90 mg.; the mean being 103.43 mg. of silver. Comparing this with the values obtained later, it shows that a dead roast is not so reliable as some other methods; besides, it involves much more labor.

V. Dead-Roasted Concentrates.

The reducing-power of the concentrates was 5.2, showing that the sulphur concentrated nearly as fast as the ore.

With the dead-roasted concentrates the charge, with borax cover, was: Ore, 0.5; PbO, 1.5; $NaHCO_3$, 0.75; SiO_2, 0.2 A-T.; and argols, 2 g. The slag was good and the pour clean. Buttons hard, clean, and copper-colored; these were scorified with 40 g. of test-lead. The cupels showed copper-stain. The silver bead weighed 114.75 mg. These results indicate that the concentration of the values is not in proportion to the concentration of the bulk or of the sulphur; hence the ore is not a good concentrating-ore.

VI. Acidic Slags.

The charge for obtaining the most acidic slag, using a borax cover, was: Ore, 0.5; PbO, 1; $NaHCO_3$, 0.75; SiO_2, 0.40 A-T.; and KNO_3, 12 g. By experiment, this is the most acid flux that it is possible to run on this ore. The charge required a long time and a very high temperature to fuse. The slag was thick, viscous, and poured poorly yet cleanly; when cold, it was reddish brown in color and stony in appearance, with distinct indications of copper oxide. The buttons were hard, matte-like, and coppery. In no sense was either slag or button good.

The buttons were scorified with 40 g. of test-lead and 0.1 g. of borax, and then rescorified without more lead. The scorifiers were badly corroded. The buttons were now small, clean, soft and malleable. Their weights were made up to 15 g. with sheet-lead, and they were then cupelled. The cupels were deeply stained with copper.

	Weight of Beads. Milligrams.	Best Five Checks. Milligrams
(*a*)	104.69	104.69
(*b*)	103.00	
(*c*)	104.19	104.19
(*d*)	104.42	104.42
(*e*)	104.43	104.43
(*f*	105.38	105.38
(*g*)	105.96	
(*h*)	107.93	
(*i*)	103.00	
Average,	104.78	104.62

The highest value was 107.93, or a difference of 3.31 mg. from the mean of the best five. The lowest value was 103, or a difference of 1.62 mg. from the mean of the best five. Too wide a variation in the results is shown for them to be satisfactory, especially taking into consideration the high heat, poor pour, long period of fusion, and double scorification.

By this method the ore gives a silver-content of 209.24 oz. per ton.

VII. Basic Slags.

The charge, using a borax cover, for obtaining the most basic slag was: Ore, 0.5; PbO, 1; $NaHCO_3$, 1.5 A-T.; and KNO_3, 12 g. This was found to be the most basic flux that could be run even approximately satisfactorily. A very slow and long fusion was required. The pour was clean and the slag moderately liquid. The cold slag was brittle, granular, stony, and gray to buff in color, with some indications of copper oxide. Buttons were hard, clean, and malleable, but extremely coppery. The buttons were scorified with 40 g. of test-lead and 0.1 g. of borax, then rescorified without further addition of test-lead. The weight was made up to 15 g. with sheet-lead, and then they were cupelled. The cupels showed strong copper-staining.

	Weight of Beads. Milligrams.	Best Four Checks. Milligrams.
(a)	102.31	
(b)	102.44	
(c)	106.98	106.98
(d)	106.95	106.95
(e)	106.17	106.17
(f)	107.19	107.19
(g)	104.80	
(h)	99.50	
(i)	99.00	
Average,	103.93	106.82

The highest value is 107.19, or a difference of 0.37 from the mean of the best four, which is a very good result. The lowest value is 99, or a difference of 7.82 from the mean of the best four, which is a very poor result. This flux shows a higher best mean and a lower general mean than the acidic flux. The variations are also greater and the slag nearly as difficult to run. The same amount of scorification is required as in the case of the acidic charge, but the clean buttons are obtained at the first fusion. Since this flux is quite as unsatisfactory as the acidic flux, some intermediate or special charge must be found for this type.

The silver-value by this method is 213.64 oz. per ton.

VIII. Use of Borax in Excess.

The charge, with borax cover, was: Ore, 0.5; PbO, 1; $NaHCO_3$, 0.75; $Na_3B_4O_7$, 1 A-T.; and KNO_3, 12 g. This charge required long, slow fusion at a medium temperature, but needed constant watching and much salting-down to prevent boiling over. The pour was good and the slag quite liquid. The cold slag was of dark purplish color and stony in appearance. The buttons were very hard and brittle and were composed mainly of matte; these were scorified with 40 g. of test-lead and 0.1 g. of borax, and rescorified with an addition of 20 g. of test-lead. The cupels showed strong copper-stains. The beads weighed as follows:

	Milligrams.
(a)	87.40
(b)	88.82
(c)	90.86
(d)	99.65
(e)	102.00
(f)	104.37
Average,	95.52

The highest value is 104.37, or a difference of 8.85 from the mean. The lowest value is 87.40, or a difference of 8.12 from the mean. These results are very unsatisfactory, and indicate clearly the disadvantages of using a large excess of borax.

The silver-values by this method are only 190.24 oz. per ton.

IX. Normal Charge for Coppery Ore.

In making up the normal charge for a coppery ore the following amounts were used, with borax cover: Ore, 0.5; PbO, 1.5; $NaHCO_3$, 0.75 A-T.; and KNO_3, 12 g. There was quick fusion at a moderate heat and a clean pour. The cold slag was stony in character, and yellowish red in color. The buttons were clean, hard, and coppery; they would not cupel directly; hence, they were scorified with 40 g. of test-lead and 0.1 g. of borax. The cupels indicated the presence of copper.

	Weight of Beads. Milligrams.	Best Five Checks. Milligrams.
(*a*)	111.00	111.00
(*b*)	111.21	111.21
(*c*)	109.38	
(*d*)	107.86	
(*e*)	109.65	
(*f*)	110.68	110.68
(*g*)	109.70	
(*h*)	108.25	
(*i*)	108.60	
(*j*)	107.80	
(*k*)	110.12	110.12
(*l*)	110.38	110.38
Average,	109.55	110.68

It is to be noted that of these results (*c*), (*e*), (*g*), (*h*), and (*i*) give a mean that checks as well within itself as the ones selected, being 109.51. The highest value is 111.21, or a difference of 0.53 from the mean of the best five. The lowest value is 107.80, or a difference of 2.48 from the mean of the best five. Although not entirely satisfactory, these results are much better than in the case of any of the preceding methods, especially when the quick, easy fusion and the clean buttons at the first fusion are considered. The results also show higher values.

The silver in the ore, according to this method, amounts to 221.36 oz. per ton.

X. Effects of Litharge Used in Excess.

With an excess of lead oxide the following charge was used, with a borax cover: Ore, 0.5; PbO, 3; $NaHCO_3$, 0.75 A.-T.; and KNO_3, 12 g. The fusion was of short duration at a low temperature. Slag was stony in character and of yellowish-brown color. Buttons were clean, soft, malleable, and cupelled direct; these were scorified with 40 g. of test-lead and 0.1 g. of borax. The cupels were coppery. The beads weighed:

	Scorification Milligrams	Direct Cupellation. Milligrams
(a)	110.28	101.03
(b)	110.48	106.72
(c)	110.34	99.90
(d)	110.62	103.45
(e)	110.95	106.55
(f)	111.04	110.00
(g)		100.02
Average,	110.62	105.24

All of the scorified buttons came within the limits of 1 in 100, with high silver-yield. The highest value, 111.04, shows a difference of only 0.42 from the mean. The lowest value, 110.28, differs from the mean by only 0.34. This excellent checking commends the use of a large excess of litharge in the assay of coppery ores.

The silver-value of the ore by this method is 221.34 oz. per ton. This is 0.02 oz. less than by the method of the normal charge for coppery ores; but the check is so much better than the last mentioned that it is far superior, while the procedure is no longer and is not more difficult. The attempt to cupel directly gave results varying from 99.90 to 110.02, hence this method is not to be recommended. The value indicated by direct cupellation is 210.46 oz. per ton.

XI. Special Scorification-Tests.

In order to determine the effects of the presence of the large amounts of niter used, a series of scorifications were run. The charge was: Ore, 0.1; Pb, 50; SiO_2, 1; $Na_2B_4O_7$, 0.5 g. These tests gave good slags and buttons that cupelled directly with only a slight copper-color on the cupel. The values were as follows:

Weight of Beads. Milligrams.		Ounces Per Ton.
(a)	21.08	210.8
(b)	21.94	219.4
(c)	21.08	210.8
(d)	21.48	214.8
(e)	21.85	218.5
(f)	21.22	212.2
(g)	21.60	216.0
(h)	21.99	219.9
(i)	21.46	214.6
(j)	21.10	211.0
Average,	21.48	214.8

These results show a lower general mean and a much less satisfactory check than either by the normal charge method or by the use of an excess of litharge. It has the further disadvantage of necessitating the working on a very small quantity of the ore. Besides, it requires as much time as either of the other methods, and hence is not so advantageous.

Comparison of Methods and Values.

Methods.	Values in Ounces Per Ton.
Dead roast,	206.86
Acidic flux,	209.24
Basic flux,	213.64
Borax in excess,	190.24
Normal charge from coppery ores,	221.36
Litharge in excess (scorification),	221 34
Litharge in excess (direct cupellation),	210.48
Scorification,	214.80

XII. Significance of Assay-Results.

From the above tabulation of results it may be inferred that:

1. A dead-roast is not so satisfactory or so accurate as a run with a large excess of niter or a scorification on a charge of 0.1 A-T., or more; moreover, the roasting involves much more labor.

2. An excess of borax causes low value-determinations, especially in the presence of large amounts of sulphur, copper, iron, or other matte-forming substances. Although it is not believed that any one with experience at the furnace would hesitate to recommend borax, if properly used, the use of large amounts approaching 1 A-T. must give low results on all types of ores.

3. An excessively acid flux fuses with great difficulty and gives buttons that are hard to handle and that are frequently lost in cleaning. Further, when the ores contain matte-forming materials, the value-determinations are low; but when very little or no matte material is present, these results are not only much too high but quite difficult to obtain.

4. A very basic flux is open to all of the objections which can be raised against the acidic flux, and in about the same ratio as regards checking-figures. However, in coppery ores, the loss is not quite so great, but in zinciferous ores the results are too high. Neither method is practical; and both give variant results most of the time, unless great care be taken in the handling of the lead-buttons.

5. Ores high in copper cannot be run by any crucible method without scorifying the lead-buttons.

6. Direct scorification on high-grade copper-bearing ores does not check as well or give as high values as a combination of the crucible and scorification-methods; besides, it is not a process which is more speedy.

7. The normal or usual charge for coppery ores gives as high results as any method on this type of ore, but its checking is not of the best as compared with that in which a large excess of litharge is used. The normal charge for ores containing some zinc gives slightly lower values than in cases in which an excess of litharge is used, but still the checks are good for the grade of ore.

8. For ores high in copper or carrying some zinc, a large excess of litharge in the charge greatly improves the buttons and renders them easily handled. Furthermore, this decreases the time and temperature of the fusion, and gives buttons that check well, with very close to actual valuations for the silver-content of the ore. The method is quick, easy, and accurate.

Diagonal-Plane Concentrating-Table.

BY S. ARTHUR KROM, PLAINFIELD, N. J.

(Wilkes-Barre Meeting, June, 1911.)

RECENT experiments indicate that the usual type of concentrating-table is not only poorly adapted to produce the desired results, but also is based upon an incorrect principle, namely, the use of riffles to perform the work of stratifying the various minerals.

We have heard a good deal about riffles for concentrating-tables; exhaustive experiments have been made to discover the proper form of riffle, or to prove the superiority of this or that form; disputes and patent-litigation have arisen over the matter of riffles, and thus many have been led to believe that the riffle is the saving-device upon which the process of concentration depends.

In the present paper it is proposed to show that the riffle is greatly overrated as to the part it performs in the concentration of minerals, and that, in the near future, it may possibly be eliminated entirely from that process.

The experiments in question have shown that the troublesome riffle can be considered of secondary importance at the most, and should be so classed in the construction of a concentrating-table built upon the right lines.

The action of any form of riffle on a concentrating-table is such as to upset and retard the process of settling and stratifying the various minerals on the table-deck.

From the deck of a riffled table no concentrates can be delivered until the deck is "bedded," that is, until a sufficient amount of metallic values has been fed to the table to spread over a large portion of the deck, forming a substratum of the heavier minerals. This substratum must be maintained by the feed within quite narrow limits, and directly proportional to the rate of discharge from the table, in order that the bed shall not be lost. On the other hand, the riffles having a very limited carrying-capacity, considerable care must be taken not to overfeed a riffled table, in which case the table proceeds, in mill parlance, to rob itself. As soon as the space between the

riffles becomes filled to and above the riffle-tops, the values then pass off the table with the tailings or lighter material, before they can be delivered by the table-motion to their proper discharge-point.

It can readily be seen that to regulate a feed that will keep a riffled table properly bedded, namely, between the points of not sufficiently bedded and over-bedded (which are not very far apart), is no easy performance from a mechanical point of view, and it is rendered doubly difficult by the varying metallic content of the pulp fed to the table.

It is not often that favorable conditions for feeding a riffled table-deck can be secured in practice; and when they are, the riffle proceeds to upset the whole business in hand. As the pulp reaches the first riffle it is forced over it by its own momentum; and this process is repeated at each succeeding riffle. In passing over each riffle, the entire mass of the pulp is disrupted and shaken up. Whatever settlement and stratification of heavy particles has taken place between the riffles, is to a large extent destroyed. Each time the pulp is forced over a riffle it is an even chance whether the lighter or the heavier particles reach the deck first. They may drop from the top of the riffle and reach the table together, in which case the lighter particles become mixed with and imbedded with the heavier, in such a manner that they are unable to stratify themselves according to their specific gravities before they again meet an obstacle to their proper settlement in the form of another riffle. In other words, the work of settling and stratifying by themselves the various mineral contents of the ore in hand, is upset and retarded as many times as there are riffles on the deck of the table.

Another serious defect in the construction of many concentrating-tables is that the line of motion of the table is parallel to the riffles upon the deck of the table, hence there is no action to counteract the downward flow of the heavier minerals and aid in their separation from the gangue matter by the dressing-water.

A series of experiments extending over a period of six years and conducted by U. S. James, of Newark, N. J., to obtain the best means of concentrating minerals by the use of a wet reciprocating-table, has resulted in a table differing radically in construction from the riffled table with parallel motion. A view of the table is given in Fig. 1.

The deck of Mr. James's table is composed of a plain, non-riffled surface, and what might be termed a slightly-riffled surface. The plain, non-riffled portion of the table is formed by two planes of different inclinations to each other, meeting in a line diagonal to the line of motion of the table. Hence Mr. James has named it The Diagonal-Plane Table-Deck. The planes in question form a basin in which the minerals of the greatest specific gravities are settled and stratified previous to their discharge upon the riffled portion of the deck.

Referring to Fig. 2, which is a plan of the table-deck, *A* and *B* are the stroke-adjustments, and *C* is the tilting-lever. The pulp, which enters the launder through pipe *D*, is fed along the upper edge of one plane and on a line parallel to the table-motion. During the travel of the pulp down the gentle incline of this plane, *G*, *G*, which may be called the receiving- and settling-plane, the heavier minerals in the pulp settle and slide gently on the surface of the plane, to the line of intersection, *H*, *H*, of the planes forming the basin. Along this line the most important action of the whole operation takes place, namely, the stoppage of the metallic portion of the pulp-flow by the rising plane, *K*, *K*, forming the lower section of the basin, and the carrying onward of the lighter or tailings portion of the pulp by the wash-water out of the basin. All these actions take place simultaneously with the discharge of the concentrates from the settling-basin by the motion of the table.

The degree of inclination of the settling-planes to each other, and the angle of their intersection to the line of the table-motion, is of the greatest importance in securing the above results.

The practice of settling and stratifying by means of a settling-basin provides for the disposition of a very wide range in quantity of metallic contents. There are no confining limits other than the limits of the basin itself. The basin does not require "bedding" and is very difficult to overfeed. It settles and discharges automatically whatever quantity of metallic particles the ore may furnish.

The riffled surface of the deck is divided into two sections, one for the reception of the concentrates, *I*, *I*, Fig. 2, and the other, *J*, *J*, for the tailings. The riffles on the concentrates

section are very thin, being but $\frac{1}{32}$ in. high. As the concentrates are discharged from the settling-basin on this portion of the deck, the low riffles allow them to spread out, which action enables such gangue as may remain in them to become free, forming a thin upper stratum, which is easily washed

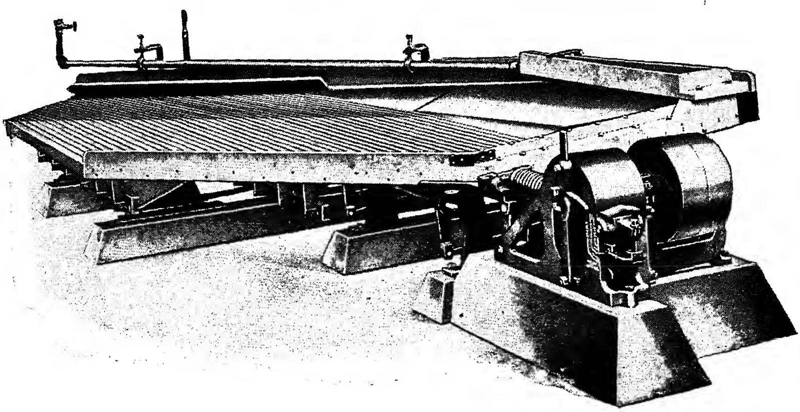
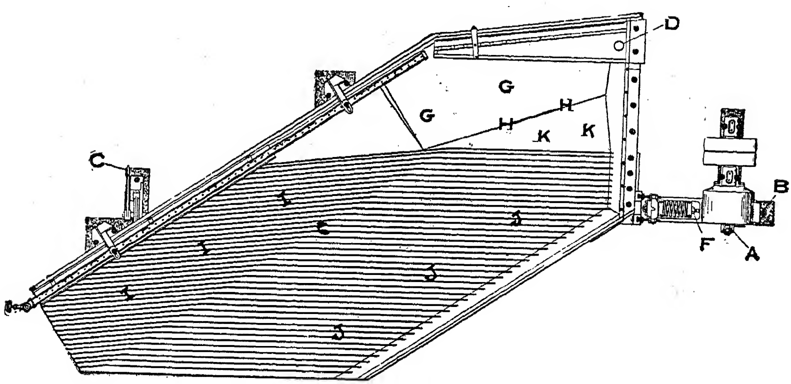


FIG. 1.—THE JAMES DIAGONAL-PLANE TABLE.



A, *B*. Stroke-adjustments.
C. Tilting-lever.
D. Feed-pipe.
G, *G*. Settling-plane.
H, *H*. Intersection of settling- and retarding-planes.
I, *I*. Concentrates-finishing section.
J, *J*. Tailings-finishing section.

FIG. 2.—PLAN OF JAMES DIAGONAL-PLANE TABLE-DECK.

away by the dressing-water. As the metallic pulp emerges from the settling-basin upon the thinly-riffled section of the deck, it is, by reason of the line of table-motion being diagonal to its line of settlement, driven not only forward but upward against the inclination of the deck, thus counteracting the

tendency of the dressing-water to wash the concentrates into the tailings-section.

The tailings from the settling-basin do not come in contact with the riffles until they have been washed entirely free of values. These riffles act simply as a retarding influence, preventing the too rapid discharge of the tailings and the wash-water from the non-riffled section. They have nothing to do with the stratification of the minerals.

In order that the table-motion may be adapted to the greatest variety of metallic pulps, the eccentric actuating the table is

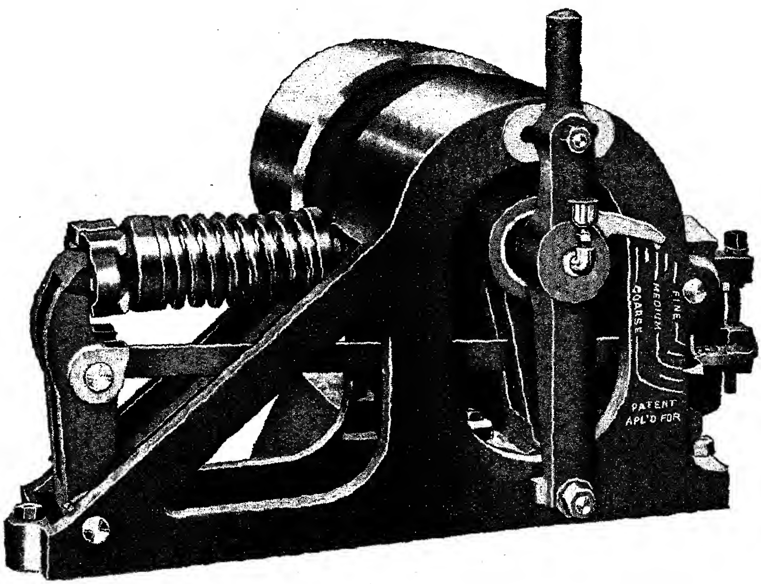


Fig. 3.—Head-Motion of James Diagonal-Plane Table, Showing Index-Plate for Regulation of Stroke.

mounted upon an adjustable pin. Thus the eccentric may be placed more or less off center. This adjustment, combined with the regular stroke-adjustment, provides for more than 200 different movements, ranging from a very mild kick and long stroke to a very sharp kick and short stroke, or any combination of movements between these extremes. As a guide to the unskilled operator, an index-plate, based upon the size of the material, is provided, which is of great assistance in selecting a movement with which to start. Fig. 3 is a view of the head-motion, showing the index-plate for the regulation of the stroke.

Electric Motors Versus Compressed-Air Engines for Driving Deep-Mine Hoists.

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(Wilkes-Barre Meeting, June, 1911)

COMPRESSED air has been and is still very extensively used in connection with mining-operations, but its application in the past has been almost entirely confined to supplying power to underground machinery. Its introduction was due to the difficulty of taking care of the steam which is exhausted underground, rather than to any advantage in efficiency to be gained by its use. The ease with which electric power may be carried to the remotest parts of the mine and the high efficiency of the electric motor have long been appreciated by mine-operators, and electric power is now extensively used underground in the more recent developments, while many of the compressed-air engines in the older workings are being rapidly replaced by electric motors.

Recently, however, it has been suggested to use compressed air for driving the large shaft-hoists, the air for the engines being supplied by electrically-driven compressors, and an installation of this character is being erected at Butte, Mont.

It is the object of this paper to compare the compressed-air system of hoisting with the various electrical systems, from the stand-points of first-cost and cost of operation. However, before entering into a discussion of the various systems, it will be well to consider briefly the conditions affecting the choice of a system of hoisting in which electricity is the ultimate source of power.

The power required by a mine-hoist is extremely intermittent and fluctuates between very wide limits, the all-day average consumption of power not exceeding from 5 to 15 per cent. of the maximum demand during hoisting.

The cost of supplying power for such a fluctuating load is necessarily higher than that for supplying an equal amount of

power delivered at a uniform rate, because of both the high first-cost and the low average efficiency of the generating, transforming, and transmitting equipment, which is only partly loaded during the greater part of the time.

Also, such an intermittent load often produces harmful fluctuations in the voltage, if the power taken by the hoist is a considerable percentage of the total load of the system or feeder circuit.

Power companies, in order to compensate for the increased cost of producing power for an intermittent or fluctuating load, usually penalize peak-loads, by making a charge for power based on the maximum demand as well as the total kilowatt-hours consumed. Also, to protect their systems from excessive fluctuations in voltage, the maximum demand is usually limited either by severely penalizing peaks over specified amounts, or by limiting the capacity of the motors which are permitted to operate intermittently on their systems.

The load factor, that is, the ratio between the average and the maximum demand, increases with the number of hoists, but the number of shafts operated by individual mining companies is in many cases so small, that equalization of the hoist-loads among themselves has little effect in reducing the capacity of the power-system serving them, or the cost of power, if power is purchased. It is, therefore, often more economical and sometimes necessary, where electricity is to be used for shaft-hoisting, to provide some means for storing power during the eriod when the demands for power are small, returning it during the peak-loads, thus limiting the demand on the power-system to approximately the average demand when hoisting at the maximum rate.

With this end in view, many systems have been proposed which take advantage of the fly-wheel, the storage-battery, or compressed air as a means of storing power when the demand for power is small, and delivering it at a high rate for short periods.

The Ilgner, converter-equalizer, and Creplet systems are the most common of the purely electric systems using the fly-wheel, and of these the Ilgner is almost universally used. Two systems using compressed air have been suggested: one, the low-pressure system, in which the hoist-engine exhausts into

the atmosphere, and the other, the dense-air system, in which the engine cylinders form a part of the closed air-system, the engine exhausting against a pressure considerably above atmospheric pressure, the admission pressure being correspondingly raised.

Not only does the choice of the purely electric system or the compressed-air system depend upon whether or not the problem presented is that of an isolated hoist or a group of hoists, but those systems which are applicable to both conditions differ in many details, depending on whether they are applied to an isolated hoist or a group of hoists. (By the term "isolated hoist" or "single hoist" is meant one which, either because of its position with respect to the power-station or other load with which its fluctuating demands for power would interfere, or because of the penalization of the peak-loads if power is purchased, must be considered as the only hoist connected to the system.)

A comparison between the purely electric system and the compressed-air system will, therefore, be made first on the basis of a single hoist, following this with a comparison on the basis of a group of hoists situated at comparatively short distances from each other, and, as both systems of compressed air are applicable to isolated installations, they will be compared with the Ilgner system, which has been almost universally adopted in the past.

The operation of hoisting-engines on the low-pressure air-system is very similar to their operation by steam. From the compressor the air is delivered to large receivers, from which it is drawn to supply the engine, the air being exhausted from the engine directly into the atmosphere. As most of the heat generated in the air by compression has been lost before it reaches the engine, it must be reheated in order to increase the efficiency, and to prevent the freezing of the moisture contained in it, which if allowed to freeze will seriously interfere with the operation of the engine.

The closed-air system differs from the low-pressure system in that the engine exhausts into a receiver, from which the air for the compressor is drawn; the working-pressure of the engine being the difference between the admission-pressure and the pressure in the exhaust-receiver.

In the Ilgner system, the hoist is driven by a direct-current motor, power for which is supplied by a fly-wheel motor-generator set. The speed and direction of rotation of the hoist-motor are controlled by varying the voltage of the direct-current generator. The maximum demand upon the power-system is maintained at approximately the average demand during hoisting at the maximum rate by automatically varying the speed of the fly-wheel, a portion of the energy of the fly-wheel being given up during the peak-loads and returned to the fly-wheel during the lighter loads.

For a description of other electrical systems, refer to a paper on Electric Mine-Hoists, by D. B. Rushmore and K. A. Pauly.[1]

At first thought the change from steam to compressed air seems to be comparatively simple, requiring much less expense than changing from steam to electricity; but a careful consideration of the problems involved will reveal the error of this first impression.

As noted above, the successful and economical operation of compressed-air engines requires the temperature of the air at admission to be such that after expansion in the cylinders its temperature is not lowered sufficiently to cause freezing of the moisture contained in it. But the permissible temperature of the air at admission is limited by the flash-point of the cylinder-lubricant, approximately from 400° to 450° F. This imposes a practical limit of 90 lb. gauge to the pressure of the air for operating the engines on the low-pressure system. Therefore, as steam hoisting-engines operate at much higher pressures, it will usually—if not always—be necessary in changing over to compressed air to replace the engines by entirely new ones having much larger cylinders, thus placing the compressed-air system on a par with the purely electrical system as far as the hoist proper is concerned.

Further, a reheater must be used with a compressed-air system. It is frequently suggested that the old steam-boilers be used as reheaters, adapting them for this purpose by rebuilding the furnaces. To do this, however, is open to two serious objections, except under special conditions: 1, with the best of care a great deal of trouble will be experienced

[1] *Trans.*, xli., 58 to 119 (1911).

from burning of the tubes; and 2, unless the nature of the fuel is such that the temperature of the air can be closely regulated, its temperature will be raised above the flashing-point of the cylinder-lubricant during the periods when the hoist is drawing no air, and disastrous results may follow the admission of this highly-heated air into the engine-cylinders. A safe way of reheating air is to inject high-pressure superheated steam in it. If the installation is a new one, a reheating-plant resembling in essential details a small boiler-plant must be built; this, however, has no counterpart in the purely electrical system.

The fly-wheel motor-generator set supplying power to the hoist-motor may readily be placed in the hoist-house or in a lean-to. The air-compressor may be placed at the hoist, or it may be included as an addition to the compressor-plant, if one exists. But, wherever placed, the building required for housing the compressor and its intercooler will be larger than that required for the motor-generator set. Further, if the compressor be placed at the main compressor-plant, rather than at the hoist, considerable expense will be involved in many cases in piping the air from the compressor to the hoist. On the other hand, if the compressor be placed at the hoist, it will be necessary to provide cooling-water, which in some cases will also involve considerable expense.

The claim is often made that the air for the hoist may be taken from an existing compressor-plant, without the necessity of materially increasing the capacity of the plant. This claim seldom, if ever, has any foundation; for if there is any considerable excess of compressor-capacity, it has been provided for a purpose, either for future growth or to take care of an emergency; and if advantage is taken of this capacity, the protection afforded by it is sacrificed, and the equivalent capacity must be added to meet the future demands.

The capacity of the air-receiver used to equalize the demand from the compressors is often underestimated in making a preliminary study of the problem. The drawing of air from the receiver to supply peak-demands for air results in a reduction of the pressure in the receivers, unless special provisions, too expensive for consideration in connection with an isolated hoist, are made to maintain constant pressure. The percentage-change

in pressure is much greater within practical operating ranges than the percentage-change in volume of air drawn, a drop in pressure from 90 to 70 lb. gauge, 22 per cent., resulting from a draft of only 14.5 per cent. of the air contained in the receiver. On the other hand, a fly-wheel gives up a large part of its energy with a small reduction in speed; approximately one-half of the total energy of the wheel being delivered with a reduction of only 30 per cent. of its speed. Therefore, the total energy stored as potential energy in the air contained in the receivers must be approximately 3.5 times that of the equivalent fly-wheel. While it is true that the drawing of power from the fly-wheel of the Ilgner system is accompanied by a loss of power, a similar loss takes place when the air is drawn from the receiver.

When lowering unbalanced or when braking, power is automatically returned to the power-system with the electric hoist. While the air-hoist engine may be made to operate as a compressor when lowering or braking, and thereby return power, it can only be made to do so at the expense of simplicity of control.

For the purpose of making a direct comparison between hoisting by compressed-air engines and by electric motors, I have assumed the following conditions.

Depth of shaft, ...	2,500 ft.
Maximum rope-speed, per minute, ...	2,200 ft.
Weight of ore per trip, ...	7,000 lb.
Weight of skip, ...	4,200 lb.
Diameter of rope, ...	1.5 in.
Time consumed hoisting ore, ...	40 per cent.
Time consumed in shifting and other hoisting, . .	20 per cent.
Time hoist idle, ...	40 per cent.
Power consumed in shifting and other hoisting, . . of that consumed in hoisting ore.	50 per cent.
Average temperature at mine, ...	45° F.
Altitude, ...	6,000 ft.
Hoist of the cylindrical-drum type, and hoisting normally done with skips in balance, but provision made in the capacity of the equipment for hoisting full load unbalanced.	
6,600-volt, three-phase, 60-cycle power available at the hoist at 1 cent per kw-hr. consumed.	
Coal having a thermal value of 12,000 B.t.u. available at the hoist at $4.50 per ton.	
Average depth of hoisting, ...	2,000 ft.

To meet these conditions with an air-hoist will require a two-stage, 6,000-cu. ft. air-compressor driven by a 1,000-h-p. synchronous motor, a storage-receiver of approximately 6,500 cu. ft. capacity, and a reheater. The equipment for the electric hoists will consist of a 550-h-p. fly-wheel motor-generator set, and a 1,000-h-p. hoist-motor.

The first-costs of these two equipments complete and installed are given in Table I.

TABLE I.—*First-Costs of Air-Hoist and of Electric Hoist.*

Air-Hoist.	
Compressor-plant, including compressors, building, and air-receivers,	$41,000
Reheater, modifications of hoisting-engine and piping,	13,500
Total,	$54,500
Electric Hoist.	
Fly-wheel motor-generator set, switch-board, and alterations to hoist-house,	$29,300
Direct-connected hoist-motor, control, and connections to hoist,	24,500
Total,	$53,800

The data in Table I. indicate that for an isolated installation the first-costs of the compressed-air hoist and the electric hoist are practically the same. In compiling these figures it has been assumed that the compressor would be placed at or near the hoist-house and only a small allowance has been made for piping from the compressor to the hoist. Of course, the variations in conditions of individual installations may cause either one or the other type of hoist to be slightly lower in first-cost, but any advantage which the compressed-air equipment may have in this respect, in individual cases, will always be insignificant when compared with the capitalization of its greater cost of operation.

The power consumed by the electric hoist and the compressed-air hoist per day for various depths and the coal consumed per day for reheating the air are shown by the curves of Fig. 1. (As stated under the assumptions made in calculating these curves, it is assumed that the power consumed in shifting and other hoisting is 50 per cent. of that consumed in hoisting ore. Therefore, if the efficiency of the two systems is figured from these curves, two-thirds of the power consumed,

as shown by the curves, should be used in making the calculations. It should also be borne in mind that the efficiency of the compressed-air system varies directly with the absolute temperature of the air entering the engine-cylinders and a corresponding correction should be made if this temperature differs from 410° F., which temperature was assumed in preparing the above figures.) These curves show that the power consumed by the compressed-air hoist is greater than that required by the electric hoist when hoisting from the greater depths, while for the shallower depths the reverse is true.

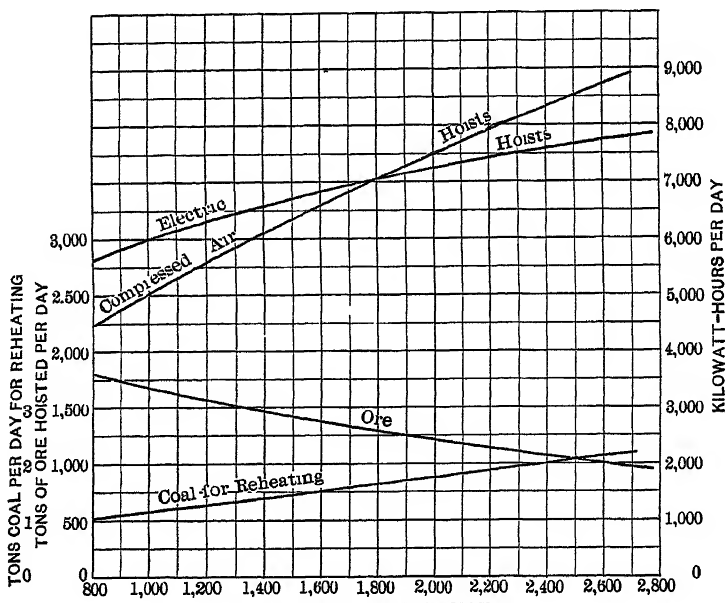

FIG. 1.—CONSUMPTION OF POWER AND COAL FOR REHEATING, ISOLATED ELECTRIC AND AIR-HOISTS.

However, these curves do not truly indicate the relative costs for power for the two systems except where power can be purchased at a flat rate, which will rarely be the case for isolated installations. The power consumed by the compressed-air hoist during hoisting is much greater than that of the electric hoist, as is indicated by the capacities of the motors driving the compressor and the generator of the motor-generator set, so that the penalization of the peak-demand for power increases the cost of power for the compressed-air hoist more

than for the electric hoist. However, as the extent to which the cost of power for the compressed-air hoist will be affected by the penalization of the peak-demand depends on the extent to which the peak is penalized, and as a flat rate for power places the compressed-air hoist in the most favorable position, I have assumed a flat rate for power in comparing the operating-costs of the two systems. Further, to the cost of power for the compressed-air system must be added the cost of reheating the air, which, although a comparatively small item, is one which cannot be neglected.

Table II. gives the total annual cost of operating the compressed-air hoist and the electric hoist based on hoisting 1,200 tons of ore per day, 275 days per year, from an average depth of 2,000 feet.

TABLE II.—*Annual Cost of Operating Compressed-Air Hoist and Electric Hoist.*

Compressed-Air Hoist.

Fixed Charges.		
Interest on investment, at 5 per cent.,	$2,725	
Depreciation,	2,180	
Taxes and insurance, at 1 per cent.,	545	
Total,		$5,450
Operating-Costs.		
Maintenance, repairs, oil, waste, and sundries,	$1,363	
Fuel for reheating air,	2,227	
Labor,	10,875	
Cost of power,	20,900	
Total,		$35,365
Grand total,		$40,815

Electric Hoist.

Fixed Charges.		
Interest on investment, at 5 per cent.,	$2,690	
Depreciation,	1,883	
Taxes and insurance, 1 per cent.	538	
Total,		$5,111
Operating-Costs.		
Maintenance, repairs, oil, waste, and sundries,	$1,076	
Labor,	4,100	
Cost of power,	19,992	
Total,		$25,168
Grand total,		$30,279
Cost of operating compressed-air hoist,		$40,815
Excess cost of operating compressed-air hoist, above that of an electric hoist,		$10,536

Table II. shows that under the conditions assumed a saving of approximately $10,500 may be realized by the adoption of the electric hoist, which saving will pay for the complete installation in less than five years.

Furthermore, a comparison of the items which comprise the operating-costs of the two systems will reveal the fact that the greater labor required for the compressed-air hoist is largely responsible for its higher operating-cost, from which it follows that for conditions differing widely from those assumed, both as to output and depth of mine, the same general conclusions may be drawn with respect to the costs of operating the two systems, namely, that the compressed-air system will be considerably more expensive to operate. This difference in operating-costs may be further increased, as previously pointed out, if the cost of power is based on the maximum demand as well as the kilowatt-hours consumed.

While it has been suggested to operate isolated deep-mine hoists on the closed-air system, the conditions rarely, if ever, are such as to warrant its adoption. For engines taking air throughout the whole or practically the whole length of the stroke, as is the case with small direct-acting pumps, small slope-hoists, etc., a considerable saving in the power required to compress the air consumed by them may be realized by adopting the closed-air system. Where the air can be and is used expansively, there is seldom any saving in power to be gained by adopting this system, and where the load varies between wide limits, as it does during a hoisting-cycle, the power required to compress the air may be even greater than with the low-pressure system. Further, the loss due to leakage will be much greater than with the low-pressure installation.

The first-cost of the complete installation will be somewhat greater than that of the low-pressure installation because of the exhaust-receiver, the return-air main, and the high pressure for which all the piping, receivers, etc., on the high-pressure side of the system must be designed.

The first-cost of the closed-air system may be further increased over that of its low-pressure competitor in special cases, due to the fact that the compressor cannot be operated in conjunction with existing compressors, and, therefore, no advantage

can be taken of the improved load-factor in reducing the total capacity of the compressor-plant.

The labor required to operate the hoist on the closed-air system will be the same as that required for the low-pressure hoist, so that the total annual operating-cost will not be very different from that given in Table II. for the low-pressure hoist. Therefore, the comparison which has been made between the electric and the low-pressure air-systems indicates approximately the saving which may be realized by the adoption of the electric hoist in preference to the compressed-air hoist operated on the closed-air system.

Where a group of hoists is to be served, the compressed-air and electric systems differ somewhat from those best adapted to taking care of isolated hoists. If the hoisting is to be done by compressed air, a central compressor-plant is placed as near the center of distribution as conditions will permit.

Where a number of hoists are operated from the same central plant their loads combine and tend to produce a uniform load; but even with a considerable number of hoists operating in conjunction, there may be fluctuations in the combined load due to the simultaneous occurrence of the maximum and minimum loads of several of the hoists. In order to take care of these fluctuations, and thereby permit of compressing the air at a rate corresponding approximately to the average demand, a large receiver for storing air is located at the central compressor-plant. It has already been noted that when air is drawn from receivers the pressure drops rapidly, and, therefore, only a small percentage of the air contained in the tanks is available for assisting in handling the peak-loads unless some means is provided whereby the pressure in the tanks is maintained independently of the quantity of air drawn. To accomplish this result, a large water-storage reservoir is placed at the proper height above the storage-tanks and connected with them, which allows water to flow into or out of the tanks as air is drawn from or delivered to them, thereby maintaining constant air-pressure in the receivers.

Where the hoists are placed at a considerable distance from the central compressor-plant it is necessary to install local receivers to reduce the excessive demands for air from the central plant during hoisting, it being more economical to provide the

local receiver-capacity than to install the large air-mains which would otherwise be necessary to prevent undue drop in pressure and loss in energy in the pipes during the maximum demands for air. The capacity of the local receivers, however, may be somewhat less than that of those required for the isolated hoist. The hoisting-engine and reheater are the same as for the isolated hoist.

Owing to the expense of the double pipe-lines necessary with the dense-air system, it is not practical to operate a group of hoists by this system, and it is, therefore, eliminated from the present comparison.

The electric system as applied to a group of hoists differs materially from that for an isolated installation. In cases where it is satisfactory to gear the motors to the hoists, alternating-current motors may be used, thus eliminating the cost of the motor-generator set. The losses in the alternating-current motor will be approximately the same as the combined losses with the direct-current motor and motor-generator set.

A central electric storage-plant is placed as near as possible to the center of distribution to provide for the fluctuations in the combined load of all the hoists, as does the central receiver and reservoir for the air-hoists. Storage-batteries connected to the alternating-current supply-system through synchronous motor-generators are used for storing electric energy during the light-load periods and delivering it during the periods when the demand for power exceeds the average demand. The storage-battery is automatically controlled in such a way that the maximum demand for power is limited to approximately the average demand, this being accomplished very simply by a relay connected in the main supply-circuit.

In some cases it may be found advantageous to use large fly-wheels in conjunction with the storage-batteries, the function of the fly-wheel being to relieve the storage-batteries of the excessive peak-loads. However, as to whether or not fly-wheels can be advantageously used will depend on local conditions, and as the use of fly-wheels will tend to reduce, rather than increase, the cost of the storage-plant, it has been assumed in what follows that storage-batteries only are used.

The hoists are driven by direct-current motors, and, as it is not necessary to make any provision at the hoists for reducing

the fluctuations in the load, such as is necessary with the air-hoists, power for the hoist-motors is supplied by direct-current generators driven by synchronous motors.

For the purpose of comparing the air-system and the electric system as applied to a group of hoists, I have assumed an installation involving a group of 10 hoists, each meeting the conditions assumed for the isolated hoist, and each located at an average distance of 1,500 ft. from the central compressor-plant. Not only does the storage-system serve to reduce the peak-loads, but it also provides power for hoisting during short periods

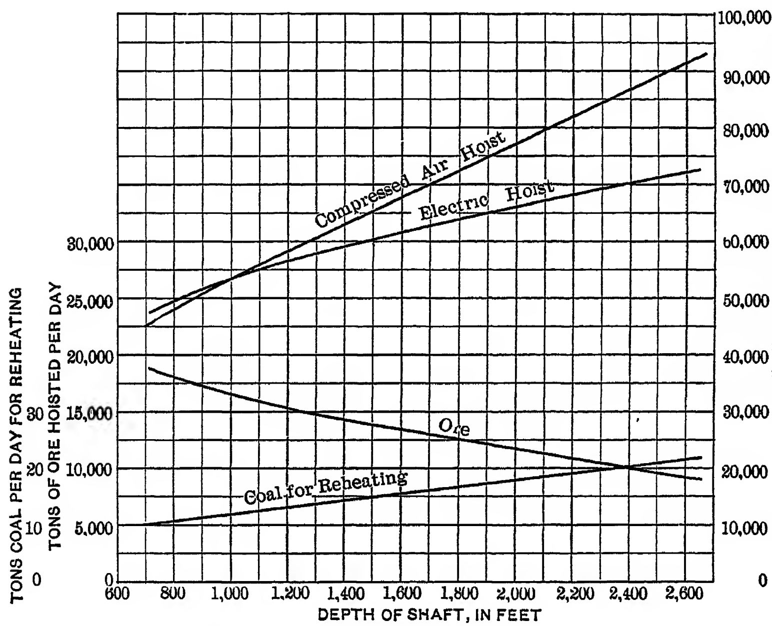

Fig. 2.—Consumption of Power and Coal for Reheating, Grouped Electric and Air-Hoists.

when the main power-supply system is shut off. It is assumed, for the purposes of this comparison, that the storage-system shall have sufficient capacity to provide power for making two trips with each hoist from the 2,000-ft. level.

To meet these conditions with air-hoists will require six 2-stage, 6,000-cu. ft. air-compressors, each driven by a 1,000-h-p. synchronous motor; a central storage-receiver of approximately 30,000 cu. ft. capacity, placed near the compressors; a reservoir of 35,000 cu. ft. capacity connected to the storage-

receiver and located approximately 210 ft. above it; and a local receiver of 5,500 cu. ft. capacity and a reheater at each hoist.

The equipment for the electric hoists will consist of an electric storage-plant equivalent in capacity to the central air-receiver and placed centrally with respect to the hoists, and a 1,000-kw. synchronous motor-generator-set and a 1,000-h-p. hoist-motor at each hoist.

The first-costs of the two equipments complete and installed are given in Table III.

TABLE III.—*First-Costs of Air-Hoists and Electric Hoists.*

Air-Hoists.		
Compressor-plant, including compressors, piping, and building,	$255,600	
Air-storage plant, including reservoir and receiver,	44,900	
Air-distribution system,	33,200	
Reheaters, local receivers, modifications of hoisting-engine and piping,	178,300	
Total,		$512,000
Electric Hoists.		
Motor-generator sets, switch-boards, and alterations to hoist-houses,	$184,100	
Electric-storage plant, including storage-battery, motor-generator, switch-board, and building,	83,700	
Distribution system, including lightning-arresters,	7,800	
Direct-connected hoist-motors, controllers, and connections to hoists,	245,000	
Total,		$520,600
First-cost of air-equipment,		$512,000
Excess cost of electric hoist above that of a compressed-air equipment,		$8,600

It will be seen from the data in Table III. that the difference between the first-costs of the electric and the compressed-air hoists is only slight; in fact, less than the variation which would be expected in making two duplicate installations. Also, with the group of hoists, as was stated for the isolated installation, the relative first-costs will vary somewhat with local conditions, but this variation will be comparatively small.

The power consumed per day in hoisting from various depths and the coal required for reheating the air are shown by curves, Fig. 2. The curves show that the power consumed by the compressed-air hoists is greater than that for the electric hoists throughout practically the whole range; and, that

the difference in the power consumed by the two systems is greater than was the case with the isolated installations. This is due to the improved efficiency of the electric hoists, resulting largely from the elimination of the friction and windage-losses of the fly-wheel and increased losses in the distributing-mains of the compressed-air system. The coal consumed for reheating the air is, as would be expected, in direct proportion to the number of hoists installed.

Table IV. gives the total annual cost of operating the compressed-air hoists and the electric hoists based on hoisting 1,200 tons of ore per day per shaft, 275 days per year, from an average depth of 2,000 feet.

TABLE IV.—*Annual Cost of Operating Compressed-Air Hoists and Electric Hoists.*

Compressed-Air Hoists.

Fixed Charges.		
Interest on investment, at 5 per cent.,	$25,600	
Depreciation, at 4 per cent.,	20,480	
Taxes and insurance, at 1 per cent.,	5,120	
Total,		$51,200
Operating-Costs.		
Maintenance, repairs, waste, oil, and sundries,	$12,800	
Fuel for reheating air,	22,270	
Labor,	74,750	
Cost of power,	212,850	
Total,		$322,670
Grand total,		$373,870

Electric Hoists.

Fixed Charges.		
Interest on investment, at 5 per cent.,	$26,030	
Depreciation—storage-battery at 18.5 per cent., remainder at 3.5 per cent.,	27,821	
Taxes and insurance, at 1 per cent.,	5,206	
Total,		$59,057
Operating-Costs.		
Maintenance and repairs—storage-battery at 6.5 per cent., remainder at 2 per cent.,	$13,292	
Labor,	38,300	
Cost of power,	180,950	
Total,		$232,542
Grand total,		$291,599
Cost of operating compressed-air hoists,		$373,870
Excess cost of operating compressed-air hoists above that of electric hoists,		$82,271

The data in Table IV. show that for a group of hoists, as was found to be the case for the isolated installations, the compressed-air hoists are much more expensive to operate than are the electric hoists, and that the saving (approximately $85,000 per year) which may be realized by the use of electric hoists will pay for the complete installation in approximately six years.

No allowance has been made in the costs of operating the compressed-air hoists for the expense of providing cooling-water for the intercoolers. In many mining-camps water is a somewhat expensive luxury, and the cost of cooling the air between the first and second stages of the compressor cannot be neglected, as it is an item which appears in full in the saving by the use of the electric hoists.

Here also, as was found to be the case for the isolated hoist, the saving in labor is the predominating factor in the total saving, and while the cost of coal and power will vary somewhat with local conditions, the results in general will be about as shown by these figures.

In determining the capacity of the storage-battery, it has been assumed that approximately 15 min. will elapse before the 20 trips (two for each hoist), which the storage-battery is designed to take care of in the event of the shutting-off of the power-supply, have been completed. This time has been allowed as under normal conditions all the hoists would not be working at their maximum rate simultaneously. If, however, conditions are such that this will not give adequate protection, this time may be shortened to 10 min. by increasing the cost of the storage-battery installations approximately $20,000.

I have purposely omitted from this paper all reference to the relative efficiencies of the systems compared, owing to the confusion which may follow the use of this term in connection with the compressed-air systems without a clear understanding of the conditions assumed in determining it. Fundamentally, efficiency is the ratio of the work performed (energy delivered to the drum-shaft) to the energy consumed in doing the work, which, for the compressed-air system, is the ratio of the work done by the hoisting-engine to the sum of the energies consumed in the compressing and reheating of the air. But as the costs of energy for compressing and reheating the air differ, no practical use can be made of the efficiency.

There is but one test of the economy of two competitive installations, and that is, cost per ton of rock hoisted; this amount including all the factors, fixed charges on the investment, operating-costs, including fuel, labor, maintenance, repairs, and cost of power. Compared on this basis the electric installation will show higher economy.

As the power required for compressing the air may be reduced by increasing the temperature to which the air is heated before entering the engine, it is often suggested that the air be heated to a temperature considerably above that assumed in this paper. To do this, however, is open to serious objections; first, because it will require the use of much more expensive engines designed along the lines of gas-engines; and, second, the reduction in the total operating-expense will be comparatively small and not sufficient to warrant experimenting with a new type of hoisting-engine.

In addition to the lower cost of operation, the purely electric systems of hoisting possess important advantages over their competitors, the compressed-air systems, which, although they cannot be capitalized, should be given careful consideration in making a choice. These advantages are:

1. The number of hoists which can be economically served from a central compressor-plant is limited, owing to the expense of piping the considerable distances.

2. The extensions may be made with less difficulty and at less expense than with the compressed-air system.

3. Simple automatic protective devices can be readily applied to the electric hoist, which not only increase its safety and reliability of operation, but also protect the hoist from abuse by a careless operator.

4. The characteristics of a hoist-motor are such that its speed is automatically limited to a predetermined value without the use of auxiliary devices, thus reducing to a minimum the possibility of a runaway when lowering unbalanced.

5. The electric system is simpler and therefore more reliable than the compressed-air system with its compressors and hoist-engines with complicated valve-mechanisms, its cooling-water system for the compressors, and reheaters for the engines.

6. The efficiency of the electrical apparatus varies little, if at all, with age, while the losses in the compressed-air system

may be materially increased in a short time by leaky valves, pistons, and air-mains, unless extreme care is taken to guard against these losses.

7. The minor repairs which are the chief sources of the annoyance in either system can be made much more quickly, and therefore with much less loss of production, with the electric hoist than with the compressed-air hoist.

In summing up, I emphasize the following:

1. That from the stand-point of first-cost, the compressed-air system and purely electric systems of hoisting are on a par.

2. That the annual cost of operating the electric hoist is much less than that of the compressed-air hoist, and that the saving which may be realized by the use of electric hoists will pay for the complete installation in from five to six years.

3. That for the isolated hoist, the maximum demand on the power-system is greater for the compressed-air hoist than for the electric hoist; and that where the peak-load is penalized, the saving which may be made by the use of the electric hoist will be considerably greater than that shown by the foregoing figures.

4. That electric motors have been used for a large part of the deep-mine hoists recently installed in Europe and South Africa, and the results obtained are such that many of the existing steam-hoists are being replaced by electric hoists, this extensive application of the electric motors throughout Europe and South Africa being sufficient testimony of their suitability for meeting conditions incident to the deep-mine hoists.

For the assistance of those who wish to investigate their hoisting-problems, I give the following brief discussion of the thermodynamics of the compressed-air engine. For a discussion of the various electrical systems of hoisting, and the method of calculating hoist-diagrams, the reader is referred to the paper, Electric Mine-Hoists, by D. B. Rushmore and K. A. Pauly.[2]

Air-Consumption of Compressed-Air Engines.—The expansion of the air in the cylinder of a reciprocating-engine follows approximately the adiabatic law, which, expressed algebraically, is:

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[2] *Trans.*, xli., 58 to 119 (1911).

$$\frac{P_1}{P_2} = \left(\frac{V_2}{V_1}\right)^k \qquad (1.)$$

Where P_1 and P_2 are the initial and final absolute pressures.
V_1 and V_2 are the initial and final volumes.
$k = 1.41$.

On the assumption that the expansion of the air in the cylinders follows the adiabatic law exactly, and that there is no rounding of the corners of the indicator-diagram due to friction and the wire-drawing in the ports during admission and cut-off, exhausting before engine has completed its stroke, and the compression due to early closing of exhaust-port, and that the back-pressure on the piston is constant throughout the stroke and equal to the external pressure against which the engine exhausts, we obtain the following relation between the mean effective pressure and the engine cut-off:

$$\text{M.E.P.} = a\,P_1 + \int_a^1 p\,dv - P_b$$

$$= P_1\left(a + \frac{a - a^k}{k-1}\right) - P_b \qquad (2.)$$

Where a = cut-off expressed in fraction of stroke.
v and p = volume and absolute pressure at any part of the stroke beyond the point of cut-off.
P_1 = the absolute pressure at admission.
P_b = the absolute pressure against which the engine exhausts.

From this equation we obtain the curves shown in Fig. 3, which give the mean effective pressures at various cut-offs for an engine taking air at 60, 100, 125, and 150 lb., and exhausting against atmospheric pressure.

Whenever reference is made to atmospheric pressure in this paper it should be understood to mean 14.7 lb. absolute, and except as otherwise stated, pressures are given as gauge-pressures.

Engines built commercially have a small clearance-space at each end of the cylinder, the effect of which on the operation

of the engine is two-fold. The mean effective pressure is higher than that corresponding to the cut-off as given by the curves in Fig. 3, and may be obtained from equation (3).

$$(\text{M.E.P.})_1 = (\text{M.E.P.}) \times (1 + c) - c\,(P_2 - P_1) \qquad (3.)$$

Where (M.E.P.) = mean effective pressure from Fig. 3, corresponding to equivalent cut-oft.

c = clearance-space (at one end of the cylinder) expressed as fraction of stroke.

When L = cut-off expressed as fraction of stroke, equivalent

$$\text{cut-off} = \frac{L - c}{1 - c}$$

Assuming 3 per cent. clearance, we obtain:

Cut-Off.	Equivalent Cut-Off.
0.05	0.0777
0.10	0.126
0.15	0.175
0.25	0.272
0.50	0.515
0.75	0.757
1.00	1.000

Further, the effect of the clearance-space is to reduce the efficiency of the engine by an amount depending on the cut-off, the maximum reduction occurring when air is taken during full stroke, the reduction becoming zero for a cut-off which allows the air to expand to atmospheric pressure before exhausting.

As pointed out, the mean effective pressures given by curves in Fig. 3 are based on the assumption that the back-pressure (P_b) on the engine-piston is constant and equal to the external pressure against which the engine exhausts. This is not strictly correct. When the expansion is such as to reduce the pressure of the air in the cylinder to atmospheric pressure, a back-pressure (P_b) of from 1 to 2 lb. is necessary to force the air out of the cylinder against the friction in the exhaust-ports and piping, and for longer cut-offs, the mean back-pressure increases with the cut-off. The mean effective pressure is further reduced by the opening and closing of the exhaust-port before the end of the stroke.

The true mean effective pressure $(M.E.P.)_2$ may be obtained from that given by equation (3) by multiplying it by a constant, depending upon the type of the engine and the cut-off. For ordinary non-condensing hoisting-engines, this constant is approximately 0.9.

The curves in Fig. 4 give the true mean effective pressures for various admission-pressures and cut-offs.

The air consumed per indicated horse-power-hour may be found from equation (4).

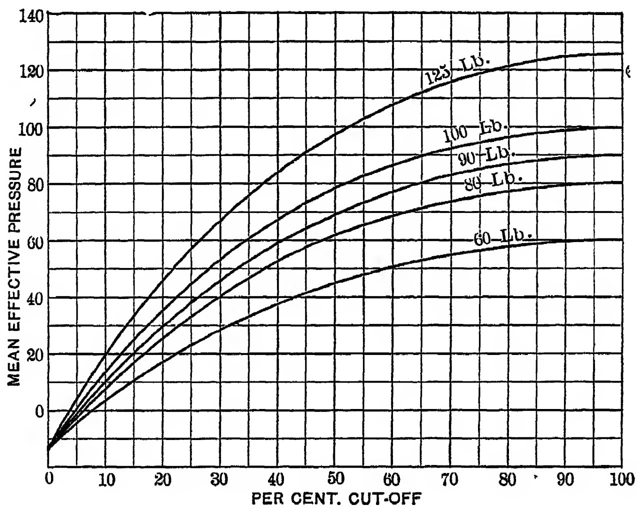

FIG. 3.—MEAN EFFECTIVE PRESSURE AT VARIOUS CUT-OFFS.

$$Q = \frac{33{,}000 \times 60 \times (a + c)}{(M.E.P.)_2 \times 144} \qquad (4.)$$

Where Q = cubic feet of air at admission-pressure.
a = cut-off expressed as fraction of stroke.
c = clearance expressed as a fraction of the stroke.
$(M.E.P.)_2$ = true mean effective pressure.

The values of Q for various cut-offs and pressures, assuming 3 per cent. clearance, are shown in Fig. 5.

The air consumed per brake horse-power-hour may be obtained from equation (4) by correcting Q for the mechanical efficiency of the engine.

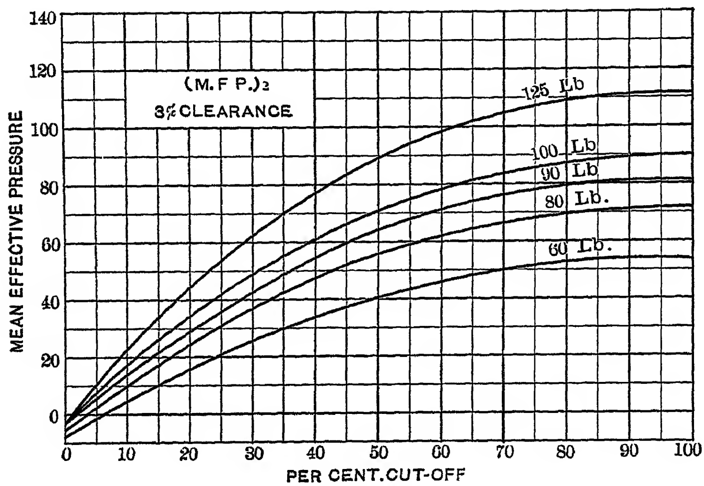

FIG. 4.—TRUE MEAN EFFECTIVE PRESSURE.

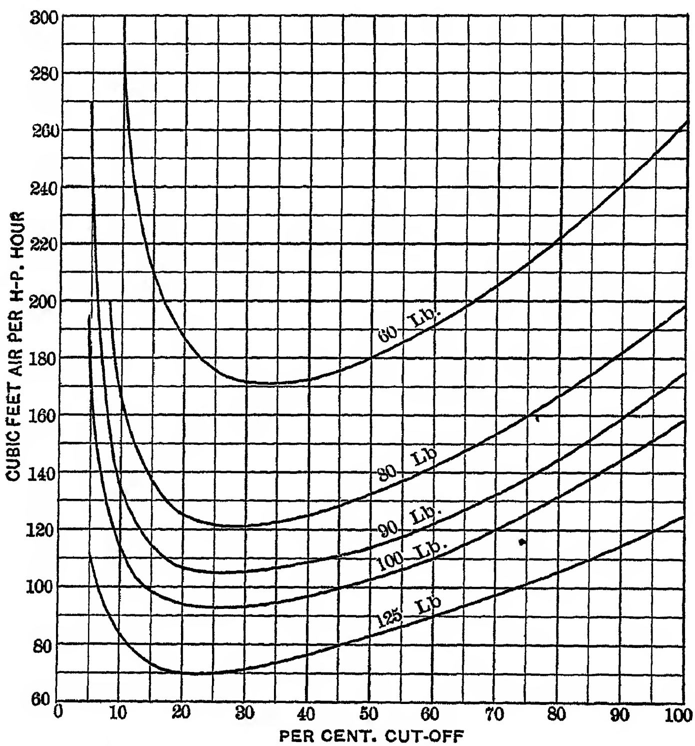

FIG. 5.—AIR-CONSUMPTION AT VARIOUS ADMISSION-PRESSURES.

Because of the low pressure-range for which air-engines must be designed, they are considerably larger than steam-engines of the same output and are, therefore, of considerably lower efficiency, this being of special importance in the case of hoisting-engines, as the torque required for hoisting in balance at full speed is usually small as compared with that required for acceleration and hoisting unbalanced, for which the engine must be designed.

As it is customary to express volumes of air in terms of cubic feet of free air (atmospheric pressure), equation (5) is given, by which the air consumed, at any temperature and pressure, may be reduced to this basis.

$$\text{Cubic feet of free air at } 32^\circ \text{ F.} = 491 \frac{Q\,P^{.707}}{T_2} \qquad (5.)$$

Where P = the absolute pressure of the air (expressed in atmospheres.)

Q = the cubic feet of air at pressure P.

T_2 = the absolute temperature, Fahrenheit, of the air at pressure P.

Power Required to Compress Air.—Without entering into a discussion of the advantages of single- and multi-stage compression, it will suffice to say that where the air cools off before it is used, it is more economical to compress it isothermally than adiabatically. It is not practical, however, to compress air isothermally, but by dividing the compression into two or three stages, depending on the pressure, it is possible to reduce the power required for compression considerably beyond that required for single-stage compression. For pressures met with in hoisting, it is customary to compress in two stages. The power required for the two-stage compression may be taken as approximately the mean between that for adiabatic and for isothermal compression.

It follows from equation (1) that the theoretical power required to compress 1 cu. ft. of air adiabatically to various pressures is expressed by the equation (6).

$$\text{Horse-power-hours} = B\left\{\int_{v_2}^{v_1} pdv + P_2\left(\frac{P_1}{P_2}\right)^{\frac{1}{k}} - P_1\right\}$$

$$= B\left\{\frac{P_1}{k-1}\left[\left(\frac{P_2}{P_1}\right)^{\frac{k-1}{k}} - 1\right] + P_2\left(\frac{P_1}{P_2}\right)^{\frac{1}{k}} - P_1\right\} \qquad (6.)$$

Where P_1 and P_2 = initial and final absolute pressures.
$k = 1.41$.
$B = 0.0000727$.
v_1 and v_2 = the volume at the beginning and end of compression.

For isothermal compression, $p_1 v_1 = p_2 v_2$, from which it follows that the theoretical energy required to compress 1 cu. ft. of air to any pressure may be obtained from equation (7).

$$\text{Horse-power-hours} = B\int_{v_2}^{v_1} pdv$$

$$= -B P_1 \log_e \frac{P_1}{P_2} \qquad (7.)$$

Assuming the compressor to draw from air at atmospheric pressure, the power required to produce 1,000 cu. ft. of air at various pressures may be obtained by equation (8) for adiabatic compression and equation (9) for isothermal compression.

$$\text{Horse-power-hours} = {}^{\circ}.0727$$

$$\left(\frac{P_2}{14.7}\right)^{.709}\left(35.9\left[\left\{\frac{P_2}{14.7}\right\}^{.291} - 1\right] + P_2\left\{\frac{14.7}{P_2}\right\}^{.709} - 14.7\right) \qquad (8.)$$

$$\text{Horse-power-hours} = 1.067\left(\frac{P_2}{14.7}\right)^{709} \text{Log}_e \frac{14.7}{P_2} \qquad (9.)$$

Equations (8) and (9) are shown graphically by the curves of Fig. 6, which also includes a similar curve for the two-stage compression.

The values obtained from equations (8) and (9) and from the curves of Fig. 6 must be corrected for losses in the compressor by multiplying them by a constant which is the reciprocal of the product of the mechanical efficiency of the com-

pressor and the efficiency of compression referred to the adiabatic. There is a further loss in compressing air, which may amount to 2 or 3 per cent., due to the moisture contained in the air.

If the temperature of the air before compression differs from 32° F., the values obtained from Fig. 6 must be corrected by multiplying them by a constant given in Table V. for various temperatures of the air entering the compressor.

TABLE V.—*Constant for Various Temperatures of Air Entering Compressor.*

Temperature Fahrenheit Degrees.	Adiabatic Compression.	Two-Stage Compression.
— 30	0.88	0.94
— 15	0.91	0.95
0	0.94	0.97
15	0.97	0.98
32	1.00	1.00
45	1.03	1.01
60	1.06	1.03
75	1.09	1.04
90	1.12	1.06

Curves of mean effective pressure, power required to compress air, etc., have not been given for the closed-air system, as a complete set of curves would be required for each pair of limiting pressures. Curves may be calculated from the preceding equations by substituting in them the proper values for P_1 and P_2.

Practical Limitations in Air-Pressure.—When air is compressed or expanded adiabatically, its temperature is raised or lowered, the relation between the pressure and temperature being expressed by equation (10).

$$\frac{P_1}{P_2} = \left(\frac{T_1}{T_2}\right)^{\frac{k}{k-1}} \qquad (10.)$$

Where P_1 and T_1 = initial absolute pressure and temperature respectively.

P_2 and T_2 = final absolute pressure and temperature respectively.

$k = 1.41.$

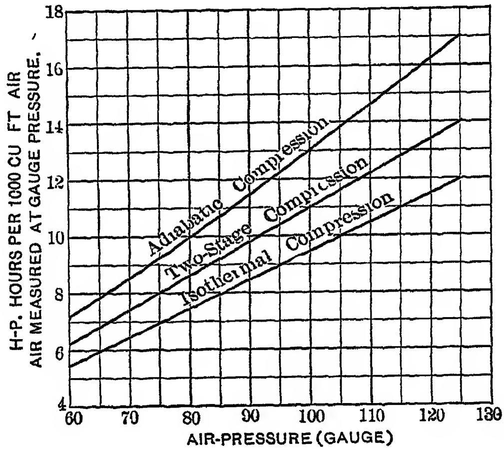

FIG. 6.—POWER REQUIRED TO PRODUCE 1,000 CU. FT. OF AIR AT VARIOUS PRESSURES. The air is measured at temperatures corresponding to pressures as shown in Fig. 7.

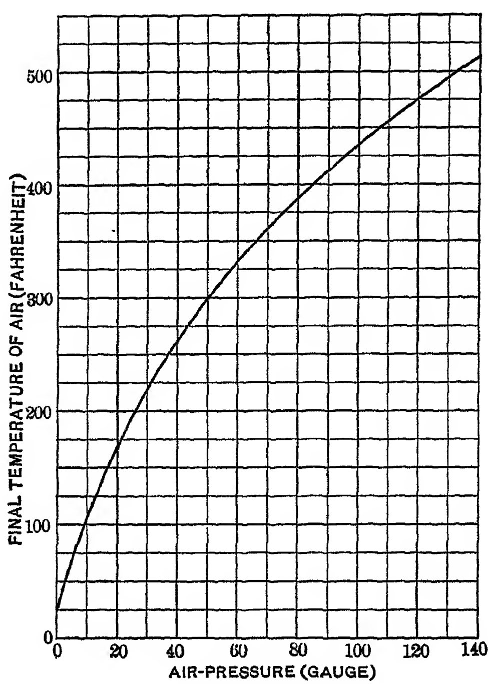

FIG. 7.—RELATION BETWEEN PRESSURE AND TEMPERATURE OF AIR.

Fig. 7 is a graphical representation of equation (10) and gives the temperature to which air will be raised by compressing it adiabatically from atmospheric pressure at 32° F. to the pressures given or the temperatures to which the compressed air must be heated, if after expanding adiabatically to atmospheric pressure its temperature is 32° F.

Now it is essential to the operation of compressed-air engines that the temperature at exhaust be kept above 32° F. in order to prevent the freezing of the moisture which is always contained in the air, unless special precautions are taken to remove it. Therefore, the temperature to which the air should be heated before being admitted to the engine varies with the pressure and the degree of expansion, that is, the cut-off, and where the cut-off varies as it does during a hoisting-cycle, satisfactory results will usually be obtained if the temperature of the air at admission is made such that after expansion to atmospheric pressure its temperature will be 32° F. It has been found by experience that with reciprocating-engines of the type commonly used for hoisting, the temperature of the steam or air at admission cannot exceed from 400° to 450° F. without causing trouble, due to the effect of the high temperature on the cylinder-lubricant. Therefore, the curve of Fig. 7 shows that the operating-pressure of the air is practically limited to 90 lb. gauge where the engine exhausts directly into the atmosphere. This limitation in pressure is a serious handicap to the air-system, because of its effect on the size and first-cost of the engine, compressor, piping, etc.

Reheating the Air.—Under practical operating-conditions, where air is stored in large receivers, and transmitted for considerable distances, all or practically all of the heat which the air contains when leaving the compressor is lost by radiation. But before admission to the engine, as previously pointed out, the temperature of the air must be raised to approximately that corresponding to its temperature after adiabatic compression. The amount of heat which must be given to the air to raise its temperature is expressed by equation (11).

$$\text{B.t.u.} = s\,(T_2 - T_1) \qquad (11.)$$

Where s = specific heat of air at constant pressure = 0.2375.

T_1 and T_2 = initial and final absolute temperatures in degrees Fahrenheit.

Three distinct types of reheaters have been proposed: 1, those in which the fuel is burned directly in the compressed air; 2, those in which the heat is applied externally, as by carrying the air through pipes over a furnace; and 3, those in which the heat is applied by injecting superheated steam into the air. The first and second methods are open to the objection that, with an engine operating on an intermittent load, the air may become overheated during periods when the engine is idle, unless automatic means, only applicable with certain kinds of fuel, are used to regulate its temperature. The third method is the least efficient, but it is not open to the objections of the first two types. The thermal efficiency of the first type may be made 90 per cent. or better, but it is questionable whether the efficiency of the second and third types exceeds 50 per cent. under actual operating-conditions. Knowing the thermal value of the fuel to be used for reheating, the quantity may be obtained by the use of equation (11), making the proper correction for the efficiency of the reheater.

Distribution of Compressed Air.—In addition to the loss by radiation of a part of the heat generated in the air by compression, there is a loss due to the frictional resistance to the passage of the air through the pipes, which loss appears as a drop in pressure. This drop in pressure due to frictional resistance may be determined from the equation developed by J. E. Johnson, Jr.[3]

$$P_1^2 - P_2^2 = \frac{0.0006\, v^2 L}{D^5} \qquad (12.)$$

Where P_1 = the absolute initial pressure in pounds.
P_2 = the absolute terminal pressure in pounds.
V = the equivalent, in cubic feet of free air per minute, of the volume of air passing through the pipe.
L = length of pipe in feet.
D = diameter of pipe in inches.

[3] *American Machinist*, vol. xxii., No. 26, p. 686 (July 27, 1899.)

Mine-Rescue Service of the State of Illinois.

BY H. H. STOEK, URBANA, ILL

(Wilkes-Barre Meeting, June, 1911.)

THE origin of the Mine-Rescue Service of the State of Illinois can be traced to two distinct sources, the work of the Rescue Station at Urbana and the Cherry disaster.

During the early part of the year 1909, the Technologic Branch of the U. S. Geological Survey, now the Bureau of Mines, in connection with the Illinois Geological Survey and the College of Engineering of the University of Illinois, established at the University of Illinois, in Urbana, a branch rescue-station to supplement the work of the Pittsburg station of the Geological Survey. As a result of the work of training at the station in Urbana by R. Y. Williams, mining engineer, and James Webb, foreman of the Bureau of Mines, and the use of the helmets at several mine-accidents in the State of Illinois, the people of the State were somewhat familiar with oxygen-helmets when the Cherry disaster occurred, in November, 1910. The oxygen-helmets were successfully used in connection with that disaster, and upon the recommendation of the Illinois Mining Investigation Commission, the Legislature of the State, assembled in special session during the winter of 1910, passed a bill appropriating $75,000 for the erection and maintenance of three rescue-stations, stipulating that they should be situated in the northern, central, and southern parts of the State. The Act also provided that the stations should be in charge of a Commission of seven, two representing the United Mine Workers of Illinois, two the mine-operators, one the Federal Bureau of Mines, one the State mine-inspectors, and one the Department of Mining Engineering of the University of Illinois.

This Commission was called together by the Governor of the State, Aug. 2, 1910, and since that time three stations have been placed, built, and equipped: at La Salle for the northern

part of the State, at Springfield for the central, and at Benton for the southern part of the State. Men are now being trained at these stations in the use of oxygen rescue-apparatus, and in rendering first aid to the injured.

Description of Buildings.—The station buildings were designed and built under the direction of the State Architect, after sketches furnished by the Commission. As the three stations were built from the same plans, a description of one building will suffice. Fig. 1 shows the Springfield Station and the three rescue-cars, before the ground about the building was graded and trees planted.

The foundations are of solid concrete. The walls of the building are of timber covered on the outside with metal lath coated with two coats of plaster throughout. The extreme dimensions are 61.5 by 87 ft. The height to the peak of the roof is 29.5 feet.

Figs. 2 and 3 show the floor-plans. The front part of the building has two floors and contains the living-apartments, office, and workshop. The rear portion contains the training-chamber, which is one story in height.

The basement contains a store-room, coal-room, and furnace-room, and has a concrete floor and finished concrete walls throughout.

On the first floor, at the left of the entrance, is the office of the superintendent, in which there is a large closet for the storage of maps. Back of the office is a hallway leading to the dining-room, which also serves as a general living-room. Off of this hall is a closet and toilet. Back of the dining-room is the kitchen, off which is a commodious pantry and a rear entrance. From the front entrance a hallway leads to the training-chamber, and on the right of this hallway is a large room used for the storage of the helmet-equipment, oxygen-tanks, potash-cartridges, and other supplies. One end of the room is fitted up as a work-shop for the repairing of apparatus, and in this part are the appliances for the charging of the electric lamps used in connection with the helmets.

The second floor includes a dormitory, containing 12 white enameled iron beds. Adjoining the dormitory is a commodious toilet fitted with lockers, shower-baths, wash-bowls, and other toilet-facilities. There is also a bath-room on the second floor.

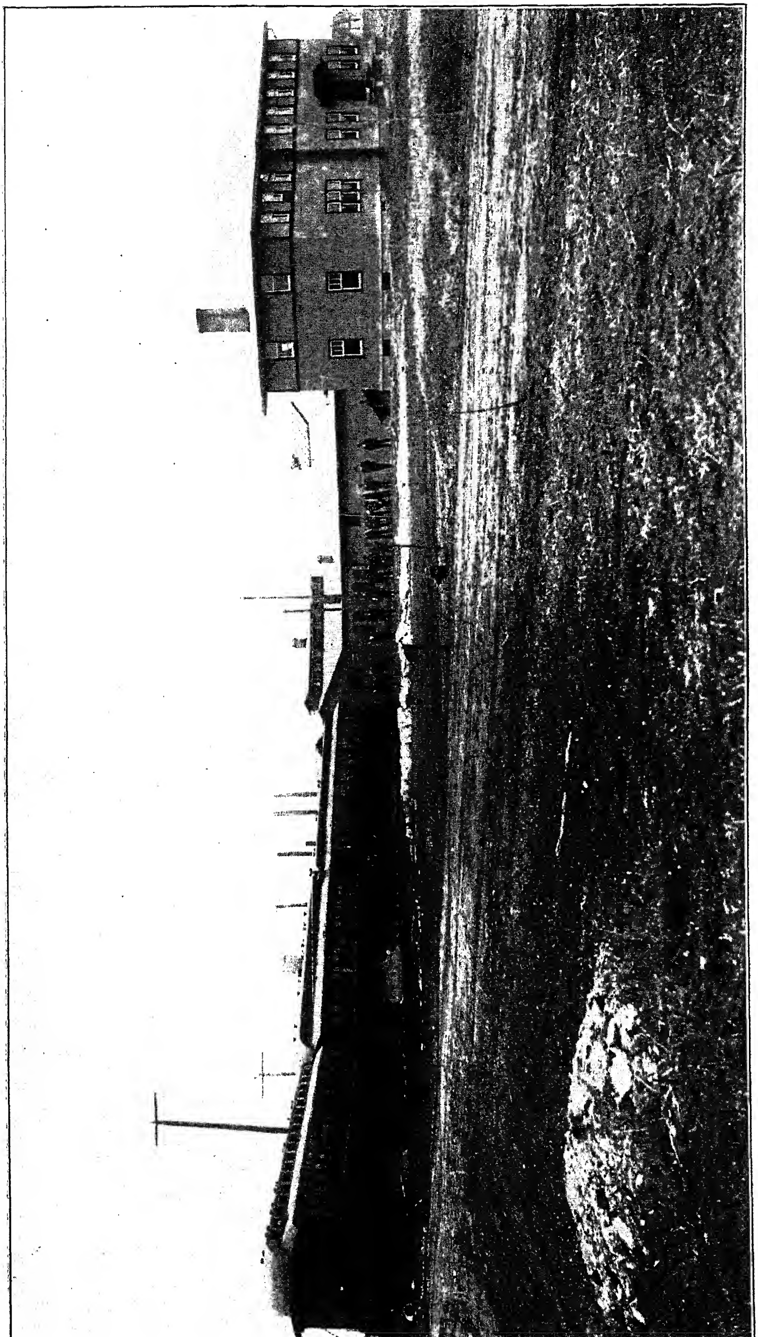

Fig. 1.—Springfield Station and Rescue-Cars.

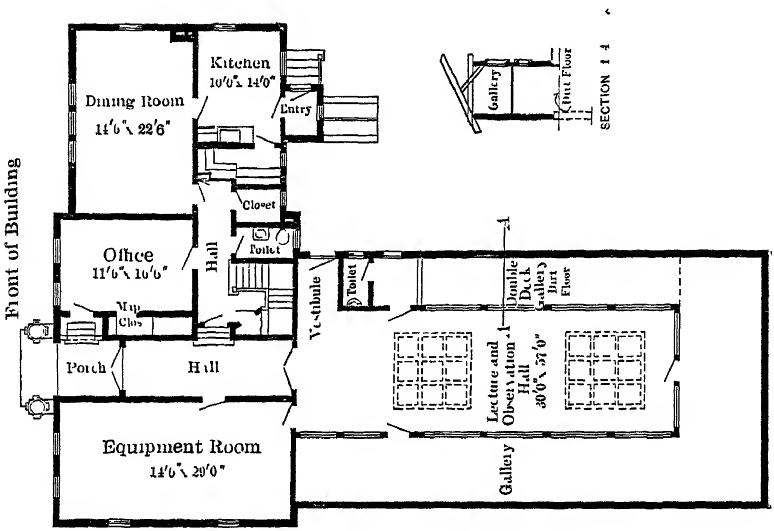

FIG. 2.—FIRST-FLOOR PLAN, SPRINGFIELD MINE-RESCUE STATION.

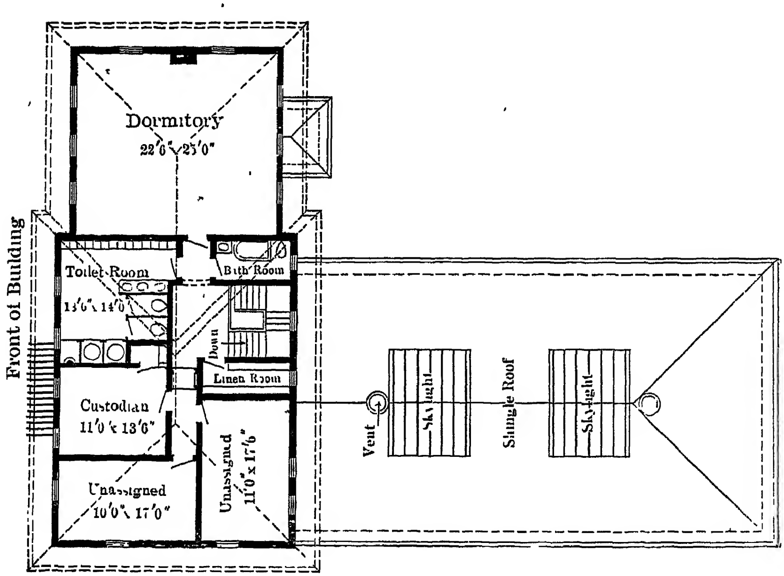

FIG. 3.—SECOND-FLOOR PLAN, SPRINGFIELD MINE-RESCUE STATION.

Three rooms are available as bed-rooms for the family of the superintendent or for other purposes. A commodious linen-closet, an attic over the front part of the building and over the training-chamber, and the cellar give ample storage-space.

The building is well lighted with electricity and thoroughly ventilated by means of numerous well-placed windows. It is finished throughout in natural wood stained a dark color, and presents an excellent appearance.

The rescue training-chamber and lecture-hall occupy the rear of the first floor. The lecture- or observation-hall is a room 30 by 57 ft., lighted from above by skylights, but it can be darkened, when desired, by curtains over the skylights. The sides of the lecture-hall are of glass, thus giving a full view of the training-gallery which surrounds the lecture-hall on three sides. The lecture-hall seats about 100 persons, is well lighted, and is provided with a special lighting-switch, so that a stereopticon can be used for lecture-demonstration purposes.

The training-gallery is an air-tight chamber in which sulphur can be burned, and in which training with the helmets and other rescue-apparatus is carried on. The right side of the gallery is 8 ft. wide and 10 ft. 4 in. high, and in this part there are placed a mine-track and a mine-car. The left side of the gallery is 6 ft. wide, and is divided horizontally into two parts, the lower part being 5 ft. 2 in. high and the upper 4 ft. 7 in. high. This division allows work to be carried on in restricted quarters, and the upper part also serves as an over-cast. In the lower part a pile of rock has been placed to represent a fall, and at one end is a toilet. The mine-track from the right side extends across the end, and there is also a tunnel through which men wearing the helmets crawl as part of the training.

Cost.—The entire cost of each building was approximately $10,000, exclusive of ground, which was donated in each city by the citizens. About each building is a commodious lot which contains a side-track for the rescue-car, and also affords space for a garden for the superintendent of the station.

Rescue-Cars.—At each of the stations there is a rescue-car for use in transporting appliances to the scene of an accident. It is also fitted up so that a rescue-party may have a comfortable place in which to stay at the scene of the accident. Two

of these cars, completely equipped, were donated to the State, one by the Chicago, Milwaukee & St. Paul railroad, the other by the Chicago & Northwestern railway. The third car was purchased from the Pullman Co., and was refitted at the shops of the Toledo, Peoria & Western railroad in Peoria, Ill.

The cars are Pullmans, and as the arrangement of the three cars is practically the same, the accompanying description of Car No. 3 will serve also for the others.

One end of the car is occupied by the heater, coal-box, and the locker for linen, and on the opposite side of the aisle is the toilet.

Three double-compartment berths on each side of the car will accommodate 12 persons, sleeping singly. The kitchen is fitted with stove, sink, and pantry, and an ice-box is beneath the car. The state-room, intended as an office for the manager or whoever is in charge of the rescue-work at the mine, contains a double berth, a desk, and a small toilet.

The end of the car used for storing the rescue-apparatus may also be used for demonstration purposes, but the space is small, and it is preferable to demonstrate the use of the apparatus outside the car, or in a suitable room. In one corner are three oxygen-tanks connected to a pump. On the opposite side is a storage-rack for seven additional oxygen-tanks. In one corner is a coal-box and in the other a locker for the pulmotor, first-aid supplies, and other small articles, and for the storage of potash-cartridges. The helmets are hung by hooks from the ceiling of the car, and, to prevent them from swinging, there is a strap that goes to the floor and is caught into a ring by a snap-hook when not in use. The helmets are covered by a canvas cover to protect them from dirt.

Helmet-Equipment.—Each station now has ten helmets of the Draeger, Westphalia, and Fleuss types, and at least five more will be added in the near future. Whether or not any one form will be adopted as a standard cannot be stated at this time; probably not; but even if this should be done, examples of other types that are in common use will be maintained at each station for purposes of demonstration.

A systematic account is being kept of the cost of operation and maintenance of the different types of apparatus used at each of the stations, but sufficient time has not yet elapsed to

establish reliable figures of cost. Since in training large numbers of men the cost of maintaining and operating a helmet is a much more serious item than the original cost of the helmet-outfit, the type of apparatus ultimately adopted for training will no doubt depend largely upon the cost of operation as determined by experiments now being made.

Each station has an adequate equipment of ordinary and electric safety-lamps, two pulmotors for resuscitation, 20 oxygen-tanks, each of 100 cu. ft. capacity, and two oxygen-pumps, one being kept at the station and one in the rescue-car, so that there is always a spare pump.

Each station is equipped with a small library of mining-books, the leading mining-magazines, and with a stereopticon. By co-operation with the mining department of the University of Illinois, lantern-slides have been furnished illustrating rescue- and first-aid work, the dangers of mining, and various other topics.

Each station has a complete equipment of supplies, charts, etc., as furnished by the First-Aid Department of the National Red Cross Society, and in the training of men first aid is of equal importance with helmet-work.

Station Staff.—According to the law establishing the Commission, the three stations are in direct charge of a manager appointed by the Mine Rescue Station Commission. Each station is in charge of a superintendent and an assistant. The salaries provided by law are as follows:

Manager, ...	$3,000 per year.
Superintendent, ...	$125 per month.
Assistant, ...	$75 per month.

An amendment appropriating money for the maintenance of the stations during the two years ending June 30, 1913, gives the Commission authority to employ such additional occasional assistants as may be needed for the operation of the cars, and for the payment of lecturers on first aid and other technical subjects.

The superintendents and assistants were selected after a preliminary competitive test and examination held at Springfield. Those who passed the preliminary examination spent several months at the Urbana station receiving training in rescue-work

and taking lectures in general mining subjects, the lectures being furnished by the staff of the Department of Mining Engineering, the State Geological Survey, and the members of the Federal Bureau of Mines located in Urbana. The men finally selected as superintendents and assistants were also given a period of training at the Pittsburg station of the Federal Bureau of Mines.

Training.—Any men, who apply to the station individually or who are sent there by their employers, are given a course of training with oxygen-helmets, in the use of oxygen reviving-apparatus, such as the pulmotor, and in first aid. When they show that they are familiar with the operation of the apparatus and can perform within a period of two hours the following tasks, they are granted certificates as members of the Illinois Mine Rescue Corps. A distinctive button is also awarded.

The tasks included in the two-hour test are:

1. Eight complete trips around gallery on ground floor.
2. Ten trips over over-cast.
3. Each man carries 25 bricks over over-cast.
4. Crawl through tunnel three times.
5. Carry four props over over-cast.
6. Saw two props.
7. Set five props and knock them out.
8. Hang canvas, take down and fold up.
9. Pull weight 60 times.
10. Two men carry dummy once around gallery, lifting dummy over car.
11. Two men push car once around gallery.
12. Eight complete trips around gallery on ground floor.

The time of training varies from one to two weeks, depending upon whether the men devote all their time to the training and live in the station during the period of training, or come to the station from adjacent mines, and devote only such time as they have from their regular duties. No charge is made for the training, and, if they desire, 12 men at a time can be lodged in the dormitory free of charge. The superintendent has the privilege of running a boarding-table for which those in training pay, or they can board outside the station if they prefer.

The mining-law passed by the Legislature recently ad-

journed provides that a map of each mine in the State shall be filed with the Manager of the Rescue Stations, and these will be kept at each station for the mines in the territory contiguous to the station, so that in case of an accident the rescue-party going from the station can study the map while *en route.*

The same law provides that candidates for the positions of mine-inspector and mine-manager must pass an examination in rescue and first-aid methods.

Although the stations have been equipped and in operation only a few months, both the operators and the miners of the State have shown their willingness to co-operate in every possible way with the Rescue Commission, and the work promises to be a potent factor not only in case of accident, but as an educational feature in combating the daily dangers of mining.

History and Geology of Ancient Gold-Fields in Turkey.

BY LEON DOMINIAN, NEW YORK, N. Y.

(Wilkes-Barre Meeting, June, 1911.)

I. Introduction.

The lack of Aryan roots for the names of metals commonly known among the Aryan settlers of Asia Minor, as well as the later colonizers of Europe, indicates that these races were generally ignorant of the use of metals until they came into contact with Semitic peoples. Practically all mining-terms in current use among the earliest Greeks resemble very strongly their distinctly Semitic equivalents, which can be traced all the way in a broad belt beginning in Lower Mesopotamia, and extending westwardly to the Syrian shores of the Mediterranean. The Greek word "metallon," for instance, used indiscriminately to designate mine or ore, probably came from the earlier Semitic equivalent, "matal." Again, the Greek words "chrysos" (gold) and "chalkos" (copper) seem to be descended from the Semitic forms "chrouts" and "chalak." It is a natural inference that primitive mining-methods were evolved by the dwellers in the mineralized areas of Asia Minor, from whom later Greek, Roman, and even North European miners obtained their first notions of the reduction of metallic ores, by

virtue of a general westward migration of mining and metallurgy. Some traces of its passage through Turkish territory will be noted in this paper.

While European Turkey can boast of one ancient gold-field, the Asiatic dominions of the Sultan may lay claim to at least two well-defined and widely-separated gold-producing districts. These three regions may be distinguished as the Thracian, the Pontic, and the Anatolian gold-fields.

II. Turkey in Europe.

1. *The Thracian Gold-Field.*

The most conspicuous topographic feature of the lowland between Constantinople and Salonica is the uplifted Archæan

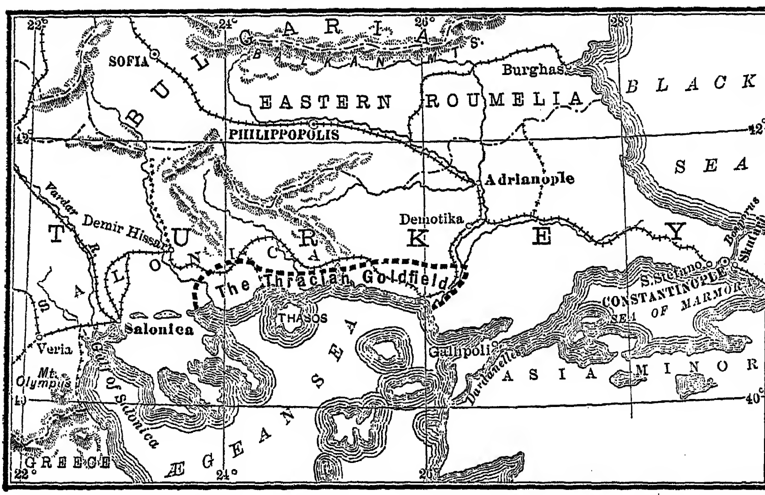

Fig. 1.—Sketch-Map of European Turkey, Showing the Thracian Gold-Field.

mass known as the Rhodope mountains. This chain appears to be a southern prolongation of the boundary-defining Kara Balkan range, from which it extends with an approximately north-south trend until it almost dips into Ægean waters at the Gulf of Lagos. It forms the backbone of the Thracian metalliferous province, and is intimately related to gold-mining in the region. Starting from within its folds, that industry found a propitious field eastward up to the site of the placers

of the Hebrus river (the modern Maritza), mentioned by Pliny.[1] On the west, gold was won as far as the banks of the Strymon[2] (the modern Struma or Karasu). These two water-courses give fairly accurate east-and-west boundaries of this important district on the mainland. The island of Thasos, lying west of the Rhodope mountains, to which it is petrologically related, also belongs to this same metalliferous province. Fig. 1 is a sketch-map of Turkey in Europe, showing the Thracian gold-field.

The Thracian coast consists of highly-metamorphosed pre-Eocene formations[3] that appear to have been much dislocated, so that the general appearance is that of an archipelago of old rocks in the Eocene sea. The component rocks include mica- and hornblende-schists, crystalline limestones and marbles, gneisses and granites, and serpentines, upon all of which Tertiary deposits rest unconformably.

The Phœnicians seem to have been the first to conduct organized mining-operations in this region. Yet there is no reason to doubt that the aboriginal Thracian tribes were acquainted with the values of the metals found in their subsoil, and it is likely that they led enterprising prospectors from the south more than once to the site of the mineral deposits, as Indians have shown quartz and other veins to the white man in the Far West. According to Greek mythological tales, mining was first undertaken on Mount Pangeum by Cadmus,[4] who settled in Thrace while engaged in his search for Europa, who had been carried off by Jupiter. Lenormant[5] claims that Cadmus in this story represents Phœnician settlers who immigrated into Thrace. The date of this beginning of what was destined to become a flourishing industry is set at 1594 B. C. by Abbé Barthélemy in "Adacharsis."[6] Other historians place it at as much as a hundred years later; but whatever be the true date, there is no doubt of the colonization of the district by Phœnician immigrants, of whom a constant procession

[1] Book xxxiii., chap. 21.

[2] J. Malcolm Maclaren, *Gold*, p. 160 (London, 1908).

[3] *Quarterly Journal of the Geological Society*, vol. lx., No. 239, p. 243 (Aug., 1904).

[4] Diodorus Siculus, Book v., chap. 48.

[5] *Premières Civilisations*, vol. ii., p. 321.

[6] W. Jacob, *An Historical Inquiry into the Production and Consumption of Precious Metals*, p. 41, footnote

from the southeastern shores of the Mediterranean was persistently wending its way northward.

The exact location of Mount Pangeum has not been established; but it is known to be in the range running parallel to the coast between the valley of Anghista or eastern portion of the valley of Serres and the high road from Orfano and Pravista.[7] It has been called Punar Dagh on some maps, and the old mine-workings are supposed to have been found on the Pilaf Tepe peak. The production of gold from this locality was large enough to give rise to various legends of the riches locked up within the bosom of these mountains. At the height of the power of the kings of Macedonia, shortly after 400 B.C., it was the prevailing popular belief in this part of Thrace that gold extracted by the pick would immediately grow again like grass mowed by the scythe.

It is not surprising that the possession of such gold-bearing lands was ardently coveted by rival Greek states. To mention but a single case, in 465 B.C., the Thracians revolted from the maritime confederacy headed by Athens, on account of a quarrel concerning the Thracian gold-mines, with the Athenian settlers at Eion, on the Strymon.[8] At that time the Thracians were actively working their own mines, although, according to Herodotus,[9] these were beginning to show signs of exhaustion. It is therefore highly probable that they were spurred on to investigate the possibilities of the adjoining mainland, and that in this pursuit, their interests clashed with those of others similarly occupied. At all events, the Thracians figure as the principal owners of the mines around Datum, a very important mining-town near the coast, and once an opulent city, thanks to the wealth which its inhabitants derived from the ownership of the gold-mines.

Another known locality of similar industrial activity lies north of Datum. It was called Crenidæ at first, and Philippi subsequently. The last name survives to this day, marking the site of ruins which the traveler cannot fail to notice, almost halfway between the town of Drama and the sea-coast. Thracian and Athenian miners had settled in this vicinity in the

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[7] Rawlinson's *Herodotus*, vol. iii., p. 219, footnote (London, 1880).

[8] Phillip Smith, *Ancient History*, vol. i., p. 457 London, 1893).

[9] E. Lenormant, *Premières Civilisations*, vol. ii., p. 331.

fifth century B.C., and for a while were very actively engaged in their craft. In 357 B.C., however, the only traces of foreign enterprise still discernible consisted of scattered abandoned workings. The mines had reverted to the Thracians, who had become effete through the distribution of wealth accumulated by their predecessors. Some time in that year, Philip, king of Macedonia, marching victoriously eastward, reached Amphipolis, 30 miles west of Crenidæ. His attention was directed to the mines, reports of the richness of which must have been still current. Probably in need of funds for the execution of his vast projects, the conquering sovereign did not disdain to investigate the old workings for himself. He descended underground,[10] and supervised in person by dim torch-light the cleaning out and unwatering of the "canals" (drifts). Canal is the term used by the Scotch historian, probably to conform to the Latin texts available to him. Pliny, throughout his *Natural History*, uses the same term to represent underground workings. Thanks to the royal initiative, the mines were soon after placed on a producing basis and the "bosom of the earth was again opened and ransacked with avidity"—according to the Scotch Historian Royal, who relies for the substance of his account on the text of Seneca.[11] It was in commemoration of this industrial revival that the town was henceforth called Philippi. The bulk of the gold extracted was coined on the spot, to the amount of nearly 1,000 talents (about $1,000,000), annually,[12] into the now exceedingly scarce Macedonian gold-pieces known to numismatists as "Philippic." This was in those days an enormous sum, having a purchasing-power far greater then than now. It bears witness to the great enterprise and activity of the Macedonians, and may also be considered as a proof of the relatively large area that must have been included in the workings, since, with the methods of extraction then in vogue, vertical depths exceeding 300 ft. must have been attained with considerable difficulty, if at all.

It is impossible to determine the length of the period of active mining-operations, after this Macedonian revival of the industry. But it seems very unlikely that Alexander should not

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[10] Gillies, *Ancient History of Greece*, vol. iv., p. 33.

[11] Gillies, *loc. cit.*

[12] Diodorus Siculus, Book xvi., chap. 8.

have followed in his father's footsteps, in fostering the industrial expansion of his empire; and we may safely assume that the mining-camp of Philippi continued to flourish for about a couple of decades, at least, during the hey-day of Macedonian supremacy. Two centuries later, after the battle of Pydus, and the defeat of Perses (about 168 B.C.), the region passed into Roman hands, and contributed its share to the periodical replenishment of the Roman treasury.[13]

In Byzantine times, these gold-mines, lying at the very door of the capital, could hardly have been overlooked by the wide-awake engineers of the Eastern Empire, whose knowledge and skill were unsurpassed in their age. When, in the third century A.D., Rome's universal but waning power, vested in Constantinople, made that the first city of the world, the gold-mines of Thrace were still furnishing large supplies of gold. Indeed, from that time to a period in the twelfth century, when Europe was deep in the gloom of the Dark Ages, it was the part of civilized Byzantium to provide a large part of the gold currency of the world, through a continuous supply of Byzantine gold coins, which found their way to the northernmost regions of the continent.[14]

Four centuries later, and about 3,000 years after this celebrated gold-field was first exploited, it happened to be visited by Dr. Belon, of Paris, a physician of Francis I. This was at the zenith of the power of the Ottoman Empire, when French statesmen were hobnobbing with their Turkish colleagues under Sultan Suleyman the Magnificent. The doctor, who was an expert in mineralogy, examined the Thracian district in 1546 and 1549,[15] and says of it:[16]

> "These mines yield so much gold and silver that the Emperor of Turkey draws from them 1,800 ducats a month, and in some months this sum attains 3,000 ducats. Within the last fifteen years the production has declined, and the duties to the Emperor have not exceeded 1,400 ducats. The persons who carried on the operations had formerly enriched themselves more than they were thought to do at present."

From his reports it appears that the mines were located on the side of a mountain in the vicinity of the village of Sidero-

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[13] Jacob, *loc. cit.*, p. 76.

[14] Finlay, *History of Greece*, vol. i., pp. 78, 167 (Oxford, 1877).

[15] Jacob, *loc. cit.*, p. 132.

[16] M. Gobet, *Les anciens minéralogistes du Royaume de France*, vol. i., p. 53.

kapso, where he found conditions similar to those which he had observed at Joachimsthal in Bohemia. The presence of a large number of miners, and the consequent opportunity for trade of many kinds, had drawn a motley gathering from all lands. His enumeration of the various nationalities assembled in that mining-camp vividly reminds us of the various races encountered to-day in any camp "out West." For their methods of mining, however, the natives had drawn on the Germans, in whose language the technical terms of operations, as well as the names of tools, were currently expressed.

2. *The Island of Thasos.*

Facing this highly-productive area on the mainland, the pile of primary rocks constituting the island of Thasos emerges out of the Ægean sea. The significant appellation of Chrysay (the Golden), bestowed upon it by the ancient Greeks,[17] shows that the fortuitous intervention of watery expanse in no wise impaired the felicitous similarity of its physical features to those prevailing on the opposite shore.

According to De Launay,[18] who has thoroughly investigated the geology of the Ægean archipelago, the island consists in the main of an extensive NW–SE. anticline of metamorphic beds stretching from the hamlet of Kasavithi on its western coast to the islet of Kynira on the east. These masses of primary rocks make up exclusively a complex of metamorphic schists, including gneisses, mica-schists, and amphibolites, with intercalated strata of crystalline limestones and marbles. Such rocks are characteristic of the Ægean region both on the European and the Asiatic shores. The metamorphosed strata strike almost due E. and W., and are very frequently horizontal. Here and there, occasional layers of recent conglomerates cap the older rocks.

By reason of the variety of minerals occurring on this island, the Thracians were famous as miners throughout antiquity. These natural resources also acted as a powerful incentive to the colonization of Thasos, as early as at least fifteen centuries before the Christian era, by the fortune-seeking Phœnicians.[19]

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[17] Arrian, Fragmenta, 67.

[18] *Annales des Mines*, Ninth Series, vol. xiii., p. 227 (1898).

[19] G. Rawlinson, *Phenecia*, p. 60 (New York, 1880).

Towards the beginning of the 5th century B.C., Herodotus's travels had taken him to the island, where he found that mining was the chief industry of the natives. Indeed, the enterprising islanders had, by this time, extended their operations to the equally rich adjoining regions on the mainland, as described above. Their annual revenue from mining amounted to 200 talents (about $240,000) in lean years, and 300 talents (about $360,000) in years of prosperity.[20] About one-fiftieth of these totals was yielded by their holdings in Thrace proper. Concerning the mines in the island, the Father of History says:[21]

"I myself have seen the mines in question; by far the most curious are those which the Phœnicians discovered at the time when they went with Thasos and colonized the island, which afterwards took its name from his. These Phœnician workings are in Thasos itself, between Cænyra and a place called Ænyra over against Samothrace; a huge mountain has been turned upside down in the search for ores."

This remarkable description seems to leave no doubt as to the exact location of these mines.[22] Yet it was impossible for De Launay[23] to detect any traces of ancient workings at the alleged site. On the other hand, he discovered ample evidence of considerable ancient labor near the hamlet of Kakiracki, built on the diametrically opposite shore. At this point, old slags had been dumped into the neighboring gulches, often filling them entirely, particularly where they lead to Sotiro. The unusually large volume of these old dumps indicated the proximity of extensive workings and their prolonged exploitation.

The inference from these two sets of observations is that two distinct periods of mining activity must have prevailed at different places in Thasos, and that all the superficial manifestations of the earlier, which obviously must be the one referred to by Herodotus, in the passage quoted above, became completely obliterated in the course of time. It should also be noted that both sites correspond to homologous points on the anticline, and that mineralization of the one would, all things being otherwise equal, warrant the assumption of a similar phenomenon at the location of the other. These facts, coupled

[20] Herodotus, Book vi., chap. 46.

[21] *Ibid.*, chap 47.

[22] This locality is probably the one called at present Kynira; it is an islet lying east of Thasos and facing Samothrace.

[23] *Loc. cit.*

with our knowledge of events in Thrace, enable us to reconstruct the story as follows:

At some time before the 15th century B.C., Phœnician explorers, sailing from the southeast, landed in Thasos at a point near Kynira, where the outcrops of the pyritic bodies (seen by De Launay) attracted their attention. That such outcrops might be auriferous is entirely in harmony with our present knowledge of this class of deposits; and the gold-bearing zone need not necessarily be confined to the mere outcrops but might comprise all the oxidized upper levels of the ore-body. The recovery of the metal would be effected mainly by means of washing and panning, although amalgamation also might have been employed occasionally, since it is now known that the properties of mercury in this connection had not escaped the attention of the ancient gold-seekers.[24] After the working of the upper levels at Kynira, and probably before any attempt had been made to invade the mainland, the surface of the island was minutely explored, and the deposit lying on its western coast was discovered and likewise made to yield its precious contents. The slags observed by De Launay indicate the use, in this district, of other metallurgical processes.

Another site of ancient exploitation is known to have existed north of Thasos, in the small island of Thassopoulos, known in the days of Herodotus as Scapte-Hyla. The annual revenue of its mines in 492 B.C. amounted to 80 talents[25] (about $100,000). One of the eminent owners of mines in this locality was the wife of Thucydides,[26] whose wealth may have enabled him to devote himself to study and literary labor.

Such is the partial record of a region, characterized by the resumption of profitable mining-operations at various intervals during nearly forty centuries. Undoubtedly much might be added by more learned and leisurely compilers to this imperfect, yet, I trust, suggestive outline. Researches into the industrial activity of former generations are not always totally devoid of economic value to the modern engineer. While many of the principles actuating ancient technical practice have now become obsolete, it may be questioned whether the

[24] Pliny, Book xxxiii., chap. 22.

[25] Herodotus, Book vi., chap. 46.

[26] Marcellin, *Vitæ Thucydide*, p. 9.

faculty of reasoning upon available data and of dealing with immediate conditions has been notably increased; and the ancients, judged according to their light and their tools, may still be worthy of our study and our respect.

No work of importance has been attempted on the mainland section of this gold-field within recent years. It is interesting to note, however, that the island of Thasos has now become a zinc-producer. The annual production of calamine from mines owned by the Metallgesellschaft of Frankfort amounts to 30,000 metric tons.[27] Whether a similar change in the metal-production of the mainland deposits will hold true, remains to be de-

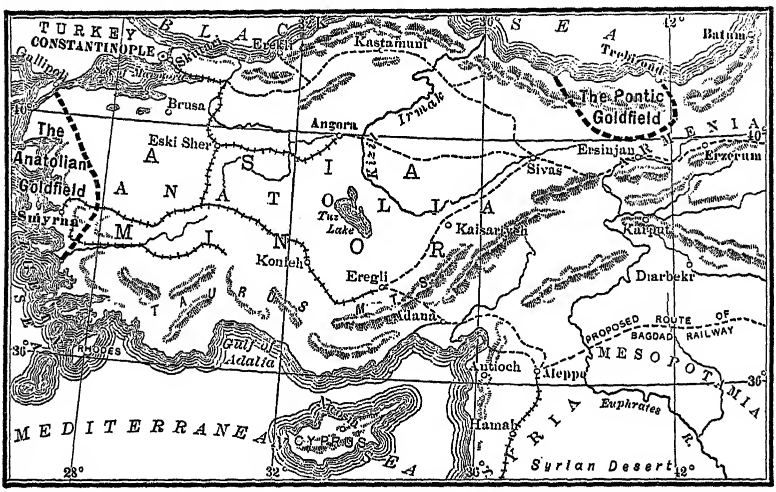

Fig. 2.—Sketch-Map of Asiatic Turkey, Showing the Anatolian and Pontic Gold-Fields.

termined by future observers; but it is quite possible, in accordance with anologies, that the future gold-production of these ore-bodies may not be again as abundant as it has been in the past.

III. Asiatic Turkey.

Three major folded arcs, forming as many independent chains of lofty peaks, fringe the wave-battered shores of Asia Minor, and, encircling, rim-like, its elevated barren plateaus, determine the trend-lines[28] of the structure of this westernmost projection of the Asiatic continent. Within the mighty folds

[27] Private correspondence. Leon Dominian.

[28] E. Nauman, *Hettner's Geographische Zeitschrift*, vol. ii., pp. 7 to 25 (1896).

of each, occurs an auriferous zone, genetically related to copious lava-flows of comparatively recent origin, detailed studies of which are yet to be made.

The Pontic gold-field lies in the most easterly, and the Anatolian gold-field in the most westerly, of these zones of disturbance, the effects of which have been so far-reaching upon the development and history of the peninsula. A third gold-field, of altogether minor historical importance, lies on the slopes of the Tauric mountains, the most imposing of these three great uplifts. Fig. 2 is a sketch-map of Asiatic Turkey, showing the gold-fields.

This occurrence, within the only zones where heavy mountain-making agencies have been at work, of the only known gold-producing areas in Asia Minor, can scarcely be regarded as a mere coincidence, though it would be hazardous, at this incipient stage of our knowledge of the geology of the region, to carry our generalizations too far.

A glance at the early history of this tramping-ground of our Aryan forefathers gives the impression that the region was both better known and better appreciated by them than by its modern inhabitants. Fully 3,000 years ago, Asia Minor, as a human habitation, was already very old, and there flourished in certain portions of it a civilization as advanced, in many of its phases, as the later Roman culture ever was.

Along with the recognition of the economic value of various ores, mining had assumed such importance as to have become the means of sustenance of numerous settlements scattered from the Ægean coastland to the Persian Gulf. Within that territory, empire after empire had risen to power, and passed into oblivion. Colonies of the vanished kingdoms of Summer and Akad, preceding the Babylonian empire itself, had flourished in the fifth millenium B.C. With the westward march of progress, the Hittite power came into being; and finally, the ten centuries immediately preceding the birth of Christ witnessed an unparalleled growth of civilization on the eastern shore of the Ægean sea. During this period Greek paganism evolved a highly-advanced organized life. In each of these successive stages of culture, the art of working ores was profitably carried on; the metals being respectively valued according to their relative abundance and usefulness, or commercial importance.

1. *The Anatolian Gold-Field.*

This metalliferous province forms part of a geologic belt extending from the plains of Troy to the valley of the Pactolus, and slightly farther south, so as to include Mount Tmolus—the modern Boz Dagh. It contributed largely to the gold-output of proto-historic times, and, as might be naturally expected, it has been duly commemorated in various legends which have descended to us, together with the superabundant exaggerations with which ancient exploits were wont to be embellished.

Its northeast portion was explored during antiquity in the vicinity of the Asiatic shores of the Dardanelles. The abundance of gold jewelry found in the excavations on the site of the several cities of Troy indicates a large production of gold from localities probably not far away. The best-known of these mining-camps of the Troad flourished between Pergamos and Ataineos, and were inhabited by the Dactyles, a hardy and enterprising race. Strabo, in the course of his travels, found numerous traces of ancient workings[29] in the vicinity of the ancient town of Astyra, then a ruined city which formed part of Abydos, but which had been independent when the gold-mines in its vicinity were productive. At the time of Strabo's visit, close to the dawn of the Christian era, the mines had been practically abandoned, and the formerly prosperous mining-camp had dwindled to commercial insignificance. The extent of the ancient workings seen by him indicates that mining had been carried on very actively at this point, and legendary tales often attribute the immense wealth of Tantalus or of Priam to the ownership of these diggings.

The site of Astyra is supposed to coincide with that of the modern hamlet of Serjiller, about 14 miles south of the Dardanelles. Abandoned workings of considerable extent are known to exist at this point, in a mica-schist country, intruded upon by lower Tertiary igneous rocks, which, according to Diller,[30] English and Flett,[31] consist of liparite, mica-hornblende, and augite-andesites, the latter in an advanced stage of decomposi-

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[29] Book xiii, chap. 1.

[30] *Quarterly Journal of the Geological Society,* vol. xxxix., No. 156, p. 627 (Nov., 1883).

[31] *Idem,* vol. lx., No. 239, p. 254, *et seq.* (Aug., 1904).

tion. All these volcanic rocks have been ultimately capped with basalt. This igneous series is remarkably similar to some which have been observed in various zones of volcanic activity within the American Great Basin region, such as the south-western portion of Nevada, where appreciable amounts of gold have been yielded by veins incased within rocks, the chief characteristic of which appears to consist in the intermediate composition, in a scale of decreasing acidity of the magmas from which they have solidified.

A portion of the large quantity of gold articles unearthed on the site of Troy must have been derived from Phrygia and Lydia, two of the most important mining-provinces of the world in the first millenium B.C. It may be recalled here that the Troad borders on Phrygia, where, according to ancient traditions, the discovery of the art of fusing metals took place in the course of a forest-fire, during which it was found that fragments of ore had been accidentally melted.[32]

There cannot be any doubt that the Phrygians, in common with their better-known eastern neighbors, the Lydians, were the most renowned miners and metallurgists during the pre-eminence of Hellenic culture. The profusion of mineral species, enumerated by Pliny as found in these kingdoms, indicates that the natives had abundant opportunities to become proficient in the arts of mining and smelting. Lydia especially was renowned for its wealthy rulers and citizens, most of whom were owners and operators of mines. Sardes, the capital, was long a world-market for gold, silver, copper, and iron. Not only did the Lydians derive large incomes directly from their underground operations, but, being situated, geographically, midway between Western culture and Eastern splendor, they managed to act as commission-agents for both parties, so that products from either direction paid them toll in transit, and thus increased the wealth of the Lydian capitalists. Herodotus mentions[33] the colossal fortune, reaching far into the tens of millions of dollars, amassed by Prince Pythios, supposed by some to have been a descendant of Crœsus, the wealthiest of the kings of Lydia. This nobleman was the dynast of Celenes

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[32] Lucretius, lines 1240 to 1243.

[33] Book vii., chap. 27 to 29.

when Xerxes invaded the West. Plutarch declares[34] that it was his custom to prevent the inhabitants of the mining-districts under his rule from pursuing their agricultural labors, lest the time thus spent be subtracted from more profitable employment at underground work. We can more easily understand such conditions when we take into consideration the great scarcity of metals, and the consequent demand for them, which existed at that time throughout Europe. The lack of gold was particularly felt in Greece in the sixth century B.C., when the Lacedemonians had to import expressly from Lydia the relatively small amount required for the gilding of a statue.[35] With regard to the wealth of Crœsus, Rawlinson, referring to Strabo, says[36] that its reality cannot be questioned; for Herodotus had himself seen the ingots of solid gold, six palms long, three broad and one deep, which to the number of 117 were laid up in the treasury at Delphi.

The height of Lydian prosperity was attained in the first quarter of the seventh century B.C., and successfully maintained during the ensuing 250 years. Throughout this period the precious metal was won both from alluvial and from deeper mining. Glowing tales concerning the gold-producing banks of the Hermos were spread to the confines of the world; and many are the legends that spring from the accounts of the rich clean-ups made by enterprising Lydian prospectors in washing the gravels of the Hermos and its tributary, the Pactolus. The latter stream owed its gold, according to an ancient story, to the fact that Midas, the mythical founder of the Phrygian kingdom, had bathed in its waters, upon the advice of Bacchus, in order to be deprived of the fatal faculty of turning everything he touched into gold. This tradition, like so many others of a kindred nature, has value only as indicating the existence of an ancient and flourishing placer-industry in the valley of the Pactolus. This river, as well as the Hermos, of which it is an affluent, rises on the northern slope of the Tmolus mountain, itself the site of numerous mining-excavations. It may be safely assumed, as an explanation of these old workings, that the discovery of nuggets in the river-sediments

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[34] *Moralité*, vol. i., p. 324.

[35] Grote, *History of Greece*, vol. ii., p. 229 (New York, 1853).

[36] *History of Herodotus*, vol. i., p. 367 (London, 1880).

stimulated a careful examination of the immediate vicinity, and that this search led the ancient prospectors to the ultimate source of the gold, namely, to the auriferous veins of the mountain.

How prolific in their yield of the precious metal these banks of the Pactolus must have been may be inferred from a partial review of the frequent allusions in ancient literature to the gold-bearing sands of this famous river. Tchihatchaff's enumeration[37] suggests the strong appeal made by this source of wealth to the imagination of ancient writers. Among others, Scylan of Caryadnis[38] speaks of the Pactolus as having formerly borne the name of Chrysoroas (the gold-bearing), by reason of its auriferous character. He claims, furthermore, that the precious element was engendered eternally in its waters. Herodotus also alludes[39] to the gold carried by this stream; and it is interesting to note that he lays special stress on the notion that the gold was primarily obtained from the flanks of Mount Tmolus. Poets and writers in endless succession have extolled the good fortune of the Lydian prospector. Virgil,[40] Juvenal,[41] Sivius Italicus,[42] all refer in glowing terms to the gold-laden muds borne along with the flowing waters. Seneca,[43] with wonted emphasis, describes the river as inundating the fields with gold (*inundat auro rura*).

Nevertheless, this production was not destined to be everlasting. In Strabo's time, at the beginning of the Christian era, it had dwindled to comparative insignificance. Philostrates[44] quotes Apollonius as saying that the Pactolus was "formerly" auriferous; and, inasmuch as this celebrated philosopher was a contemporary of Nero and of Vespasian, it may be inferred that very little gold was recovered from this source at that time. The same writer advances the hypothesis of the primary derivation of the nuggets from the very rocks of Mount Tmolus, and his assertions in this respect indicate a remarkable

[37] Asie Mineure, *Géog. Phys.*, vol. i., p. 240.
[38] Apud Hudson, vol. i., p. 14, *et seq.*
[39] Book i., chap. 93, 101.
[40] *Æneid*, Book x., line 142.
[41] *Saturnalia*, Book xix , line 298.
[42] Book i., line 158, 234.
[43] *Phœnissis*, line 604.
[44] *Appolonius Tyannis*, Book vi., chap. 57.

soundness of deductive reasoning. In the light of modern theories on placer-formation, a part of their metallic contents may well have been derived from the rocks incasing the veins which, in the course of their erosion, have contributed the bulk of the metal subsequently re-deposited in the form of nuggets.

A later writer, Festus Avenius,[45] makes use of the term "auriger" in the text of a description of this affluent of the Hermos. His use of this adjective need not, however, be taken as indicative of a renewed activity of mining on the Pactolus. It may have been employed by way of reminiscence only. Such, indeed, appears to be the case in the writings of Constantine Manasses,[46] a Byzantine writer of the eleventh century; and John the Lydian,[47] a native of the valley of the Hermos, alludes to the Pactolus merely to refer to its past contributions to the world's wealth. In our own time, peasants dwelling in the vicinity of the Boz Dagh are known to make a scanty livelihood by washing the gravels brought down by the rivers. But their appearance and mode of living are far from supporting a belief in the continued abundance of the yellow metal in that region. It is therefore possible that the placers of this gold-field were exhausted fifteen centuries ago, although the same assertion might not be made with regard to the original sources of the nuggets discovered by the ancients.

The ambition of these early Greek miners was not confined to alluvial mining. Numerous deeper workings have been found on the slopes of Mount Tmolus. Farther north and in a similar direction from the bay of Smyrna, similar vestiges of ancient labors are to be seen on Mount Sipylus—the modern Manissa Dagh. Thomae,[48] speaking of gold-ores in the vilayet of Aidin, refers to this locality as the one from which part of the wealth of Crœsus was derived. He says that the ancient workings had not been fully fathomed, although a vertical depth of 200 ft. below the crown of the hill had been reached. The same observer calls the country-rock in these mines a trachyte, which he found to be very much decomposed in the

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[45] Apud Hudson, *Descriptio Orbis Terræ.*

[46] *Compendium Chronicum,* line 6258.

[47] *De Magistratibus Populi Romani,* Book iii., p. 258.

[48] *Trans.*, xxxviii, 222 (1898).

upper levels, worked by the Lydians. Small veins, cutting across the same volcanic rock, were found to carry argentiferous galena, blende, copper, and iron pyrites with gold, all with a quartz gangue. An average sample, taken from a 1- to 2-ton lot of the ore, assayed as follows: Gold, 13 dwt., and silver, 5 oz. 13 dwt. Troy per ton; lead, 7.6, copper, 2.2, and zinc, 2.7 per cent.

The Lydians could fairly claim to be the first users of coins in history. This, in itself, bespeaks the abundance of the precious metals in that richly-endowed country. It was quite natural that accumulations of gold and silver should eventually be bartered for commodities brought from all over the world to this meeting-point of the East and the West. To stamp the metals with distinctive signs, and use them as a measure of value, was the next step, and an easy one in the ordinary course of commercial transactions.

The earliest products of the Lydian mints were issued during the seventh century B.C.; and were made, not of pure gold or silver, but of a compound of both, known as "elektron," in which the ratio of gold to silver was four to one by weight. The name is supposed to be derived from the identical Greek word, designating amber, which the native alloys of those metals somewhat resemble in color. A century later, gold and silver coins appeared; and, no doubt, this change was associated with the discovery of a method of parting the two metals. Gold and silver generally occur in nature in alloys of various proportions, the character of which is particularly evident where the veins containing them are the ultimate manifestations of volcanic activity. The Anatolian gold-field, for instance, belongs to such a region of vulcanism, where gold-bearing veins, occurring in igneous rocks, carry a noteworthy amount of silver. But, apart from all extreme manifestations, the general phenomenon is, that metallic gold occurs in nature generally alloyed with silver (and not with copper). So universal and so well-recognized is this phenomenon, that the distinguished mineralogist, Breithaupt, Professor of that science at Freiberg, classified native gold and native silver as one species, ranging in composition from gold with a trace of silver to silver with a trace of gold, and denied the occurrence in nature of either metal without some alloy of the other. The

proportions of the two metals in native alloys vary with the composition of the minerals from which they have been reduced. It seems probable, therefore, that the "elektron" of the Lydians was simply the native alloy characteristic of their own district, and was adopted for coinage and commerce until the discovery of a method of parting permitted the manufacture of gold and silver coins separately.

2. *The Pontic Gold-Field.*

In the northeastern portion of Asiatic Turkey, and at the point of junction of three empires, the snow-capped peak of a huge Tertiary volcano, familiarly known as Mount Ararat, rising in majestic loneliness above all surrounding eminences, marks the center of a region characterized by repeated volcanic eruptions, and the point of intersection of two main axes of high uplift. One of the latter sweeps westwardly, to form a long mountain chain which borders all the northeastern shore of Asia Minor, and within which gold-mining has been actively carried on since proto-historic times.

An interesting clue to these very ancient operations is afforded by the text of a portion of the second chapter of Genesis (vv. 10–12):

> "And a river went out of Eden to water the gardens; and from thence it was parted, and became four heads.
>
> "The name of the first is Pison: that is it which compasseth the whole land of Havilah, where there is gold;
>
> "And the gold of that land is good; there is bedellium and the onyx stone."

By many Bible students, the river Pison has been identified as the modern Tchoruksu, running generally parallel to the east-west extension of the coast. Its valley has been since time immemorial a region of exceeding fertility, and has also enjoyed, thanks to the sheltering barrier formed by the elevated Pontic range along the northern bank of the river, the added blessing of immunity from the ravages of the bleak northern gales of Russia. It is not surprising that the combination of such advantages awakened desire for their possession in ambitious leaders of different periods; and many are the tales of struggle and bloodshed over the ownership of these gold-fields.

One of these stories is repeated by Strabo,[49] whose explora-

[49] Book xi., chap. 14, 19.

tions of the then known world, at a time when traveling was beset with innumerable difficulties, have made his name illustrious among students of the geography of antiquity. It appears that Alexander the Great, perhaps remembering his father's successful mining-ventures in Macedonia, received intimations of the abundance of gold in the Sambana district, which lay in the province of Syspiritides (the modern Izpir), within the Pontic productive area. Straightway he dispatched Menon, one of his generals, at the head of an armed force, commissioning him to secure possession of the wealth-yielding territory. The sturdy natives, however, resisted the great conqueror's designs regarding lands which they justly regarded as their own, and having routed the invaders, sent back to Alexander the head of Menon, his general.

Some eight centuries later, gold-mines south of the harbor of Trebizond, in the same district, became the subject of dispute between Justinian, the mighty Byzantine emperor, and Chosroes, the King of Persia, his foe.[50] At that time the workings, operated on a very extensive scale, were furnishing abundant supplies of the precious metal for the mint at Constantinople. Much of this gold was won from placers along the banks of the Tchoruksu and its tributaries, the latter having their sources in the southern facets of the Pontic range.

Strabo's copious notes here become again instructive.[51] He says that the natives recover gold by first straining the auriferous muds through screens and subsequently spreading the undersize over sheepskins specially selected on account of their long fleece, the shreds of which would serve to entangle the particles of metal. Incidentally, it may be noted that the derivation of the appellation "Land of the Golden Fleece," by which this northeastern portion of Asiatic Turkey was designated in the oldest of the tales of Greek mythology, becomes self-suggestive. The corroborative testimony supplemented by the name of Cape Jason, applied to a nearby promontory, tends to remove all shadow of doubt regarding the exact location of that once-famous Eldorado.

The period of its original discovery, however, cannot be determined as closely as its location. The earliest known record

[50] Gibbon, *Decline and Fall of the Roman Empire*, vol. iii., p. 579.

[51] Book xi., chap. 2.

is the mythical narrative of the Argonauts in search of the Golden Fleece; and this story yields but a single credible fact —namely, that, at some time in early Greek history, not unlikely about 1000 B.C., yet perhaps a few centuries later, a band of adventurous Greek emigrants decided to set forth and discover the country from which they had received from time to time reports of the existence of untold wealth in various forms.

There is no doubt that, from that time on, and far into the fifth century B.C., the various Greek communities were actively engaged in the exploration and colonization of the regions lying east of their mainland. Such expansions in the course of a national growth have invariably been the consequence of prosperity at home. It is not inconceivable that some of the hardier and more indefatigable of these explorers surmounted the hardships attending travel on the turbulent waters of the Black sea, and succeeded in reaching portions of its south-eastern shores. What they saw there may be inferred from the tales which they brought back, enriched with the adornments required to fire the imaginations of their countrymen.

According to the version of Pliny,[52] Strabo's younger contemporary, and one of the best known naturalists of antiquity, the Colchis, as he calls the Land of the Golden Fleece, was ruled, previous to the coming of the Argonauts, by Selances, a descendant of Actes. This ruler is said to have discovered extensive gold-placers in the territory inhabited by the Suanes, who lived within the pale of the Colchides. "The whole country, however, is renowned for its gold-fields," is Pliny's final comment in connection with this description.

IV. Prospects of the Future.

To our own generation the point of greatest interest in connection with any of these gold-fields lies in the possibility of a resumption of exploitation of the hitherto abandoned workings. This does not necessarily imply that gold will again be the chief metal recovered. There have been numerous instances where mines, at one time gold-producing, have eventually turned out to be great producers of copper. Two noteworthy instances of such a sequence are furnished by two of the

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[52] Book xxiii., chap. 15.

world's largest present deposits of low-grade copper sulphides: the Mount Lyell mine in Tasmania, and the Rio Tinto in the Spanish province of Huelva. The former came into prominence in 1881, and began to attract attention as a gold-producer in the incipient stage of its development.[53] With regard to the latter, Strabo, to whom frequent reference must perforce be made in connection with ancient mining, has given us an enthusiastic account of the gold-production in southern Spain on the site of what are now the famous and immensely productive copper-mines of Rio Tinto.

Another instance of the same nature occurs at the Mount Morgan mine in Australia. Here the ore at very shallow depths was rich in gold and carried only insignificant quantities of copper. Lower down, however, the percentage of the latter metal grew considerably higher.

There are some signs of the recurrence of the same phenomenon in the Pontic gold-field. Copper has been mined during the past few centuries at various points within this metalliferous province. Although these operations have been desultory, there is ground to suspect the existence of a rich copper-belt parallel with the northeastern coastal development of Turkey in Asia. Kerassons is, among others, a noteworthy locality in which copper-ores in large bodies have been reported on various occasions.[54] The recovery of gold as a by-product in the smelting of such ores is by no means impossible.

Work on the Anatolian gold-field, on the other hand, has remained practically at a standstill since the beginning of the Christian era. Perhaps detailed investigation of the region will lead to interesting industrial developments; and, while these ancient gold-fields may never again yield such quantities of the precious metal as they gave to the miners of antiquity, they may produce, through development at lower depths, of the baser metals, a greater treasure than they conferred on former generations.

[53] *Engineering and Mining Journal*, vol. lxxxix., No. 14, p. 713 (Apr. 2, 1910).
[54] *Mining and Scientific Press*, vol. xcviii., No. 24, p. 821 (June 12, 1909).

Treatment of Nicaraguan Gold-Ores.

BY HENRY B. KAEDING,* PIS PIS, NICARAGUA, C. A.

(Wilkes-Barre Meeting, June, 1911.)

Introduction.

THIS paper presents the results of experiments in the treatment of the gold-bearing ores of the Pis Pis district, near the Atlantic coast of Nicaragua, C. A.

Up to the present time, the methods in use in this section of the country for the extraction of the values from the ores have been of the crudest, and the waste has been criminal in its enormity. The transportation of heavy machinery being difficult and costly, recourse has been had to flimsy and inadequate installations, involving great wastes in operation. In some places $15 ore is considered the lowest workable grade. One mine that came under my observation had a 7-ft. vein of $23 ore when examined for purchase; it has been running for years and is now deeply in debt.

The future metallurgy of these ores must depend upon chemical, mechanical, and economical considerations—the last being, of course, a resultant of the two former, discussed in the light of prevailing or realizable commercial and industrial conditions.

I. CHEMICAL.

The ores consist principally of quartz, carrying galena, pyrite, marcasite, and chalcopyrite, and occasionally magnetite, hematite, pyromorphite, and sphalerite, in which minerals the gold lies. The quartz veins occur between walls of andesite, "porphyry," or dioritic rocks, occasionally limestone and dacite. Often the dacite itself is heavily ribbed and filled with auriferous quartz, and is mined as ore.

Owing to the excessive rainfall of the country, the ores above water-level have been oxidized and leached of their sulphides, and have become in that operation highly acid and free from

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copper. Below water-level they are sulphides, and require an entirely different treatment.

Since the surface-ores have been about exhausted in the known mines, a brief description of the practice of the past will suffice. The ore has been crushed either under stamps or in Huntington mills, the pulp run over amalgamating-plates, the slimes thrown into the adjacent creek, and the sands leached with cyanide. The extraction on the plates has averaged about 47 per cent., the cyaniding of the sands another 10 per cent., and the slimes and tailings about 43 per cent., which went down the creek. Caustic soda has been used to furnish the necessary alkalinity, and sodium cyanide, containing a certain amount of sodium hydrate, has been used instead of potassium cyanide. The result has been that the sodium hydrate has reacted on the acid solutions of iron and alumina contained in the ores, producing a colloidal gelatinous precipitate of the hydrates of the two elements, which immediately diffused itself throughout the slime in suspension in the pulp, and not only prevented it from settling or curdling, but readily passed through a filtering medium until in its passage it had filled the pores, after which no filtration could be obtained. In consequence of this experience, it has been generally accepted as an axiom that the slimes of these ores cannot be cyanided and filtered. One slime-plant has been installed, using the old decantation process,[1] but is not a success. At this mine, however, P. A. O'Brien has lately been conducting extensive experiments, with the result that he has installed with perfect success a modern filter-plant, using it for the filtration of the slimes, while the sands are leached.

No attempt has been made to prevent the formation of the colloidal iron and aluminum hydrates, and much effort and money has been wasted in a futile endeavor to settle them, or filter them, after they were formed. So the slimes have been thrown away, and their gold with them.

The sulphide ores below the water-level have been severely let alone. Since, in most of the properties, the sulphides must be treated somehow at once, if the business is to go on, the problem becomes a vital one.

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[1] *Trans.*, xli., 998 (1911).

After a series of experiments on the ores of the Siempre Viva mine, I find that chemical success in treating this ore depends on fine grinding, while success in filtering depends upon the rigid exclusion of sodium hydrate from the process. A pulp consisting of 1 part of slime to 6 parts of either water or cyanide solution, rendered alkaline with lime, will settle in 15 hr., so that crystal-clear solution can be drawn from the surface, and the resultant pulp will have a ratio of 2 : 1. Moreover, this pulp will filter with great rapidity on a Butters leaf. I obtained a perfect cake $\frac{7}{8}$ in. thick in the remarkably short space of 1.5 min. I have tried hindered-settling, and de-watering in cones, without success, as the slightest vibration destroys the settling; dead settling in vats is necessary.

I find, furthermore, that by grinding the heavy sulphide ores so that 90 per cent. passes 200-mesh, and agitating with lime and potassium cyanide, I obtain an extraction of 90 per cent. of the values in less than 24 hr. For these tests, I used the basest ore to be had, so as to secure the most adverse conditions.

The consumption of cyanide was 2.9 lb., and of lime 4 lb., per ton of ore treated.

After confirming these experiments on larger lots of ore, I am warranted in saying that the best metallurgical method for the treatment of both the oxidized and the sulphide ores of this mine is by fine grinding and cyaniding, without previous amalgamation, in a solution of potassium cyanide rendered alkaline with lime, and the subsequent filtration and washing of the resultant slime.

Precipitation by zinc, either as shavings or as dust, is entirely successful. There is no field here for electrolytic precipitation with its cumbersome equipment and resultant base lead-bullion.

As the bullion contains a high base-content, principally copper, and as the export tax is upon the gross ounce of bullion and not upon the fine-gold content, it would appear that a cheap method of bullion-refining should be installed. The abundance of hydro-electric power would suggest an electrolytic method. T. W. Bouchelle, head metallurgist of the Lone Star mine, is working along these lines.

While I have experimented with the ore of the Siempre

Viva mine only, I believe that the same processes of treatment will apply to the ores of the other mines in the district, which are similar in all respects to this.

II. Mechanical.

The preparation of the ore for cyaniding will vary in different localities with the character of the ore. In mines where the oxidized surface-ores have not been entirely worked out, the material will probably go to the mill wet, and, to a great extent, sticky. These gummy ores can be best prepared for fine grinding by breaking in either a Blake or a Sturtevant roll-jaw crusher, and reducing to $\frac{1}{8}$ in. under stamps, or in a Huntington mill. The sulphide ores can best be prepared by passing first a Blake-type breaker, and then either under stamps or between rolls to $\frac{1}{8}$ in. In either case, after being reduced to $\frac{1}{8}$-in. size, the product should be led to a classifier, and the sands from this should be ground in a tube-mill or in grinding-pans until from 80 to 90 per cent. will pass 200-mesh. The secondary crushing and the grinding should be done in a solution of potassium cyanide, rendered alkaline with lime, the strength of the solution in either of these chemicals being determined for each particular ore. It may be found advisable to use lead acetate or oxide, in cases where soluble sulphides occur in the ore.

The tube-mill product should either be sent to a second classifier or returned to the first, the sands again going to the tube-mill and the slimes-overflow all going to settling-vats, set at a sufficient distance from the crushing-plant to escape vibration. After from 15 to 20 hr. settling, and decanting of clear solution, which for safety and additional clarification should be run through a sand-clarifying tank, the thickened slime may be drawn to the agitators for additional agitation, thence to storage-tank and filter; the residues going to the creek, and the filtered solution to the sand-clarifying tank and extractor-house.

III. Economical.

Under this head may be taken up the question of the class of machinery best suited to the obtaining of the above results, both chemical and mechanical.

In view of the miserable roads, cut until almost impassable by the rains, the feet of oxen and other beasts of burden, the freight on large pieces of machinery is prohibitory. Hence the use of such machinery as can be carried in sections is the first thing to be thought of. Rock-breakers and rolls of all classes are built in sections; the heavier parts of Huntington mills, even, may be dragged in on sleds. All the parts of stamp-mills are capable of being freighted in—except the mortars, which must come in sections; and a sectional mortar is not satisfactory. However, it is only with regard to the fine-grinding machinery that much difficulty will be encountered. A tube-mill, as a whole or even in large sections, is out of the question. The choice therefore rests between a sectional tube-mill, a sectional Hardinge mill, and grinding-pans. In considering the tube-mill or Hardinge mill, we find that the cost of importing silex for linings and pebbles for grinding will make the cost of grinding in this type of mill quite high. If, however, a quartz could be found near the mine, sufficiently hard to be used in a lining of the El Oro type, and to furnish pebbles, then I would consider these mills the best for the purpose. The grinding-pans would become economical only under certain special conditions of environment.

The question of power to drive the machinery does not enter into this calculation at all, for the reason that most of the mines own their own hydraulic power.

IV. Costs.

Potassium cyanide costs here about 24 cents; lime, 10 cents; and zinc, 14.5 cents per lb. Labor-cost, \$1.25 a day; power, practically nothing. Under these conditions, at a plant treating 100 tons per day of 24 hr., the cost of treatment from ore-bin to mint should not exceed \$1.50, and will probably be found to be nearer \$1.20 per ton.

The Continuous System of Cyaniding in Pachuca Tanks.

BY HUNTINGTON ADAMS, NATIVIDAD, OAXACA, MEXICO.

(Wilkes-Barre Meeting, June, 1911.)

THE arrangement of a flow of cyanide-pulp through Pachuca tanks in agitation, so as to permit a continuous process, instead of alternate filling, agitation, and emptying, has been proposed by various writers within the last two years, and more particularly by A. T. Grothe, agent for the Brown patents on Pachuca tanks in Mexico. It was first put into practice, I believe, by M. H. Kuryla at the Esperanza mine in El Oro, Mexico.

The starting of agitation in Pachuca tanks after filling may offer no serious difficulties with ores which do not settle rapidly in such tall tanks; and the adaptation of the tanks to continuous agitation under such conditions may give simply a somewhat more convenient method of treatment and greater agitation-capacity for a given number of tanks, because of the saving of time lost in filling and discharging. But in the treatment of pulp which tends to settle rapidly, as in the cyaniding of concentrates or of the whole pulp of ores containing heavy sulphides, the packing of the slime at the bottom may cause much trouble. The use of the radial air-pipe attachments near the top of the cone and of the air-valve outside of the air-lift tube at the bottom may obviate the difficulty to some extent; but the action of the radial air-pipes on the cone-sides is that of a sand-blast, and their continuous use cuts through the tanks. Moreover, even when these pipes are used, some pulps will pack tightly below them in the cones. Under such conditions the use of Pachuca tanks with intermittent filling and discharging becomes a troublesome process. Time is lost in starting agitation; large quantities of compressed air are wasted; pulp is blown over the tank-tops; and not infrequently it may be necessary to dig out the bottoms of tanks by hand. This trouble is the only important one occurring in the use of Pachuca tanks; and, since it is caused by intermittent filling, the arrangement of a continuous flow of pulp from tank to tank,

kept always full and in agitation, offers a means of avoiding such losses and of making the process much more satisfactory.

For a continuous flow of pulp from tank to tank, the outflow must be equal to the inflow in each tank, that the level may remain constant; and if the inflow is mixed thoroughly with the pulp already in agitation in the tank, as it would be in the central air-lift, then, roughly speaking, that part of the inflowing pulp which flows out of a tank in a short period of time will be to the whole inflow in that time as the quantity of inflowing pulp is to the whole charge. Thus, if a tank contain 100 tons of pulp, and 10 tons flow in during an hour, roughly, one-tenth of the latter, or 1 ton, will flow out to the next tank in the first hour; one-tenth of the ton which flows into the second tank will pass to the third, and so on through the series. The number of tanks in the series will, therefore, determine the power to which the fraction is raised for a short period of time. As the process is continuous, obviously these figures are not exact; but for practical purposes we may assume that with a series of tanks, the part of the pulp receiving a shorter period of agitation than the average will be balanced by the part receiving a longer period, and that in a series of six tanks having a capacity of 600 tons in all, and with 10 tons an hour passing through the system, the pulp would receive 60 hours' agitation. The same tanks, if filled, agitated, and discharged by the intermittent system, would give only 40 hours' agitation.

As the thorough mixing of the pulp in the tanks takes place in the central air-lift tubes, the overflow-connections from one tank to the following should be arranged so as to sample the overflow of the air-lift. This sampling should make a cut of the whole thickness of the stream of pulp from the air-lift. Failure in this respect would lead to classification in the tank, which would prevent the consistency of the pulp remaining the same through the whole series, and cause a thickening or a thinning that would interfere with the smooth running of the process. Any arrangement for a continuous system should also be provided with by-pass connections, so that any tank or tanks may be thrown out of the series when necessary, to allow for changes or repairs in the air-valves or interior piping, which are subject to much wear, or for any accident which may occur,

such as the dropping of a tool into a tank; otherwise, costly shut-downs and the emptying of the whole series must occur from time to time.

A. T. Grothe[1] has proposed an arrangement for continuous agitation. The overflow-connections consist of straight piping at an inclination of 60°, having the intake in each tank at a point midway from the central air-lift tube to the tank-side at two-thirds the height of the tank, and the discharge into the succeeding tank at the top of the cone. The pipe-intake in one tank is joined to the discharge in the next by a piece of rubber hose. By-pass arrangements do not seem to have been provided, and the pipe-inlets are placed far below the pulp-surfaces.

M. H. Kuryla[2] installed continuous agitation in a roughly similar form at the Esperanza mine. The tanks are 45 ft. high and 14 ft. 10 in. in diameter. The pipe-connections have their inlets 2 ft. from the 15-in. air-lift tubes and 7 ft. below the tank-tops (5 ft. 3 in. to 3 ft. 3 in. below the pulp-levels), and their discharges just below the tops of the cones in the succeeding tanks. Valves and piping are provided for by-passing any tanks in the series at half the height of the tanks, and for compressed air to clear out the connecting-pipes, in order to prevent their clogging with slime.

Both the above systems are arranged with the inflow-openings of the pipe-connections in the form of pipe-ends far below the pulp-surfaces, which have the defect of not being so placed as to assure a good sampling of the contents of the tanks.

At the Natividad mine, Ixtlan, Oaxaca, Mexico, the 100-ton cyanide-plant has been equipped with continuous agitation in a different form.

The ore contains from 5 to 8 per cent. of the sulphides pyrite, galena, and blende, and the value is chiefly in gold occurring in the pyrite. The low value of these sulphides when concentrated to 10 per cent. insoluble, and the high freight- and treatment-costs, make inadvisable the shipment of concentrates if a fair extraction can be made by cyaniding them. Tests on cyaniding the concentrates showed an extraction of from 92 to 93 per cent. of the gold, and 90 per cent. of the total value, if ground fine

[1] *Mexican Mining Journal*, vol. xi., No. 2, pp. 2 to 5 (Aug., 1910).
[2] *Idem*, pp. 44 to 46.

enough. The mill, as first put into commission in January, 1910, was not equipped for concentration. The ore was as nearly all slimed as practicable in the ordinary way with tube-mills, and the overflow from the Dorr classifiers (of which 90 to 95 per cent. passed a 200-mesh sieve) went to Pachuca agitators after thickening in Dorr thickeners. Recently, Johnston vanners have been added, to concentrate the slime-overflow from the Dorr classifiers before the pulp passes to the agitation; and the concentrates (90 per cent. through 200-mesh) are now returned to the tube-mills for regrinding, and circulate from the tables through the tube-mills and classifiers back to the tables, until so fine as not to be caught among the concentrates,—after the method of F. C. Brown.[3]

While the pulp from the above dressing, even before the addition of the vanners, was probably as fine as any usually agitated in Pachuca tanks, nevertheless, during the filling of the tanks, a part of the pulp settled rapidly to the bottom, while the lighter part, containing schistose gangue, showed very little clear solution above it, if left to settle quietly for 6 hr. Though the Pachuca tanks used are smaller than those commonly installed in Mexico, being 12 ft. in diameter and 35 ft. in height, the starting of agitation after filling commonly gave such difficulty as to cause several hours' delay, during which, at intervals, the compressed air had to be shut off from other tanks in agitation, in order to raise the pressure to 60 or 100 lb. so as to blow out the settled slime at the bottom of the tank to be started. Frequently, it was necessary to make use of hydraulic force from a pipe-line of 500 ft. head (installed originally for another purpose) to force an opening through the bottom of the settled pulp. But for the help of this latter force, the intermittent agitation would have involved the frequent digging out of tanks by hand.

In Kuryla's installation at the Esperanza mine, although he used roughly diagonal pipe-connections from tank to tank, the last tank was arranged to discharge from the overflow of the central air-lift tube into a box a little below its top, in order to gain head in passing to thickening-tanks before filtering. Since the object of the connections should be to sample the stream overflowing from the air-lifts, the box-arrangement

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[3] *Mining and Scientific Press*, vol. ci., No. 9, p. 273 (Aug. 27, 1910).

at Esperanza is nearer the desired form than the submerged pipes, and naturally suggested a similar arrangement for the whole series of tanks.

This arrangement was carried out at Natividad for the series of tanks, with the exception of the last, as shown in Fig. 1. A drop of 4 in. is used from tank to tank, and the central air-lift tubes are cut down or added to, in order to give this drop. Wooden boxes 7 in. wide, 10 in. long, and 6 in. deep (inside measurement) are fixed against the 15-in. central tubes, with their tops flush with those of the tubes, and 4-in. pipes, placed horizontally, pass from the bottom of each box to the next tank in the series. By-pass pipes, fitted with valves, join each pipe-connection with the next in the series, as shown in the plan, and the 4-in. drop from tank to tank is made in the by-passes. Since the level of the inflow-pipe in each tank is 2 in. below the tops of the air-lift tube and of the overflow-box connected with the outflow-pipe, but is slightly above the pulp-level in the tank, no part of the pulp entering can pass out of the tank without having first gravitated to the bottom, and risen through the air-lift tube, thoroughly mixed with the whole content of the tank. On the tops of the overflow-boxes are sliding iron covers, which open at right angles to the direction of the overflowing stream. The regulation of the flow from tank to tank is done entirely by means of these covers, and the valves are used only when it is desired to by-pass tanks. The boxes are sufficiently large for the tonnage passing through agitation, so that the covers need be opened only an inch or two, in order to give the required flow. The openings which sample the pulp-stream are thus rectangular, with their long axes parallel to the radial overflow at those points; and, while theoretically this is not as correct a shape to sample the stream as would be a sector of a circular ring, in practice it has been found to offer no difficulties, while it is simpler to install and to keep in order in the pulp-stream.

As the last tank in the series discharges to the pulp-tank of a Moore filter, where an intermittent feed is important, it is arranged with two pairs of sliding doors on the central air-lift tube, so as to permit agitation at various levels. The pulp is drawn off intermittently to the filter through the bottom discharge-opening of the tank, either by hydrostatic pressure in

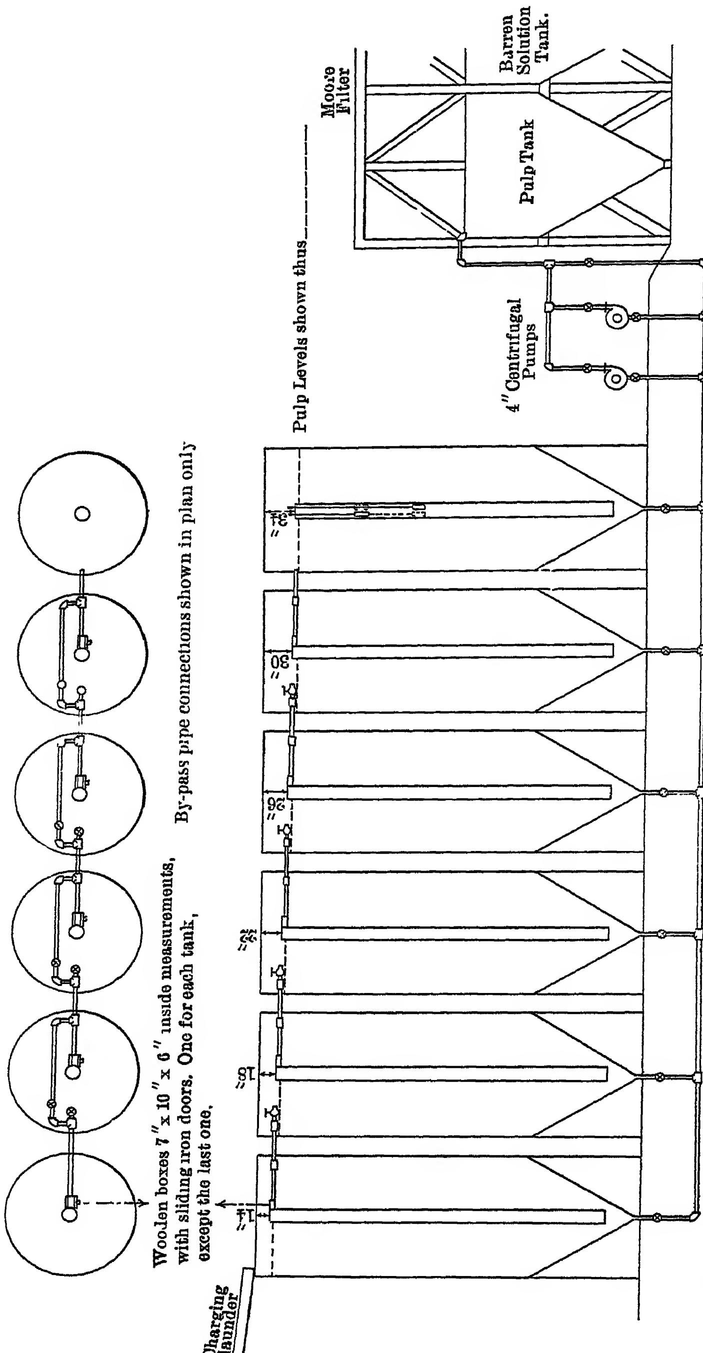

FIG. 1.—PACHUCA TANKS ARRANGED FOR CONTINUOUS OPERATION AT THE NATIVIDAD MINE, IXTLAN, OAXACA, MEXICO. PLAN, AND LONGITUDINAL VERTICAL SECTION.

the tank, if full enough, or by 4-in. Butters centrifugal slime-pumps.

All the tanks retain their bottom discharge-connections to the centrifugal pumps which lift to the filter, so that whenever it is necessary to empty any of them, they may be cut out of the series, to prevent fresh pulp entering, and after sufficient agitation, may be discharged to the filter.

In practice, the continuous system of agitation has removed completely the former difficulties in starting tanks, and gives greater capacity or longer agitation for the same tonnage. This is probably the cause of the better extraction noted by Kuryla at Esperanza with continuous agitation. While a better extraction is probably gained at Natividad for the same reason, it has not been readily measurable in practice, because slime-concentration for regrinding of the sulphides, as described above, was commenced simultaneously with the continuous agitation and caused a gain in extraction which obscures the slight gain there might be because of the change in the agitation arrangement. The continuous system of agitation has been in constant service since September, 1910, without having shown appreciable classification, though slight daily differences in the proportions of the pulp from tank to tank are caused by changes in the feed. The flow from tank to tank gives no difficulty, and is regulated by tank-boys at a cost of 62 cents American currency per 24 hr. The by-pass arrangements are very satisfactory, and any one or several tanks can be thrown out of the series whenever necessary.

The advantages of the arrangement for continuous agitation at Natividad over those of Grothe and Kuryla seem to be:

Greater simplicity of installation (the whole change from the intermittent to the continuous system was made in four days); greater accessibility for handling and supervision (all connections are above the pulp-levels of the tanks and within reach from the main deck on top of the tanks); no plugging of the pipe-connections can occur, and no compressed-air connections are necessary to free them; and as the boxes, by which the flow from tank to tank is regulated, make a good sampling-cut of the thoroughly mixed pulp overflowing from the air-lift tubes, no classification is liable to occur, and the proportion of solution to slime remains the same throughout the whole series of agitators.

Notes on Huntington Mills in Nicaragua.

BY CLARENCE CARLETON SEMPLE, NEW YORK, N. Y.

(Wilkes-Barre Meeting, June, 1911.)

AT a number of mines in eastern Nicaragua, 3.5- or 5-ft. Huntington mills are used for grinding gold-ore after a preliminary breaking in jaw-crushers. The smaller mills are made sectional to facilitate transportation to the more difficultly accessible mines. The capacity of the smaller mills is about 20 or 25 per cent. that of the larger, but in proportion to the quantity of ore ground they require more power and are subject to greater wear. The notes herein given are from records of the performance of the larger mills at one of the mines, but, in general, apply to the smaller mills also.

THE NATURE OF THE ORE.

The ore occurred as a series of flat veins in a highly-decomposed rock, probably alaskite. The primary mineralization comprised pyrite, pyrrhotite, marcasite, chalcopyrite, and quartz; the oxidation-products of which were chiefly hydrated iron oxide or limonite, chrysocolla, azurite, and malachite. The kaolinization of the feldspar of the country resulted in the admixture of a great quantity of clay with the oxidized ore. The quartz was sandy, imbedded in the clay; and probably the greater part represented the residue from the crushed rock in the fissures after leaching, and kaolinization of the feldspar. There was little vein-quartz in the unaltered primary sulphide ore. Only the oxidized ore was mined for the gold that it contained in amounts that averaged $4 per ton. No specks of gold were ever seen in the ore, and the gold obtained by panning was always exceedingly fine. The amalgamation-bullion was about 800 fine in gold and from 60 to 80 in silver. The ore delivered at the mill contained from 5 to 15 per cent. of moisture; during the rainy season the quantity was much greater. The moisture and clay made the ore so sticky that it had to be raked over steeply-inclined grizzlies; it hung in the ore-bins,

and clogged the feeders to an annoying degree. Even the pulp with 8 or 10 parts of water to 1 of solids banked on launders placed at usual grades, although there were practically no heavy minerals in the ore.

The Treatment of the Ore.

The ore, delivered at the mill in sizes up to 8 or 10 in. in diameter, was dumped upon grizzlies spaced 1 in. apart. The large lumps were spalled, and, with the grizzly oversize, broken in a 9- by 15-in. Blake or an 8- by 12-in. Dodge crusher, the broken ore falling to a 350-ton bin to join the grizzly undersize. The ore was fed to five 5-ft. Huntington mills by Challenge feeders. The pulp from the mills passed over amalgamation-tables each 4 by 25 ft., thence passed without further treatment to the tailings-pond. Plain copper plates were used that originally had been dressed with a small quantity of zinc-silver-gold amalgam.

Brass wire cloth, 30-mesh, No. 30 wire, screens were used in the mills. A sizing-test of the pulp showed 12 per cent. through 30- on 40-mesh; 8 per cent. on 60-; 12 per cent. on 80-; 15 per cent. on 100-, and 53 per cent. through 100-mesh. Of the pulp that passed a 100-mesh screen, about 20 per cent. was finer than 200-mesh, and about 10 per cent. remained suspended in water after the pulp had been shaken and allowed to stand 20 sec. Although amalgamation-tests demonstrated that 54 per cent. of the gold was amalgamable, an average recovery of but 33 per cent. could be obtained on the plates, due in part to the rapid coating of the plates by the hydrated iron oxide and clay, necessitating frequent dressing; to the admixture of lubricating-oil and grease from the mills; to the necessity of employing unskilled natives as amalgamators; and also because much of the gold that escaped amalgamation was in the coarse sand and could have been saved in amalgamation only by finer grinding of that portion of the pulp.

Power, Capacity, and Speed of Mills.

The power for the mill was supplied by two plain horizontal slide-valve engines—one of 40, the other of 60 h-p. The smaller engine drove an 8- by 12-in. Dodge crusher, two Chal-

lenge feeders, two Huntington mills, and two 10- by 54-in. Frenier sand-pumps; the larger, a 9- by 15-in. Blake crusher, three Huntington mills, and their feeders, and two sand-pumps. About 12 h-p. was required to drive each Huntington mill; but this varied with the load and speed. The boiler-capacity of the plant was insufficient, but when a gauge-pressure of 125 lb. could be maintained, the mills made 90 rev. per min. and ground from 70 to 75 tons of ore per 24 hr. The usual mill-speed was 70 rev. per min. and the capacity about 50 tons per 24 hr. The large capacity of the mills was due to the softness of the ore, much of it requiring disintegration rather than crushing, while the largest pieces, constituting not less than 20 per cent. of the feed, were 1 in. in diameter.

The mills were most efficient when running at the higher speeds. Although in excess of the speed recommended by the manufacturers, it was found that by driving at 90 instead of 70 rev. per min. the capacity of the mills was greatly increased, with relatively small increase in power. It was also noted that better recovery was effected by the amalgamation-plates when the mills were running fast. The wear on parts was relatively less at high speeds; and no more troubles or serious consequences from accidents were experienced at 90 than at 70 revolutions.

Inside Amalgamation Not Always Successful.

Amalgamation inside the mills was not successful; while 640 oz. of amalgam was taken from the outside plates, only 1 oz. was recovered from the mercury-well, *G*, Fig. 2, in the bottom of the mill. At some of the Nicaraguan mines where Huntington mills are used a good recovery is effected in the mill; but in this instance it was a failure, due no doubt to the extreme fineness of the gold. No scrapers were used, as it was found that the mullers swept all the ore and water to the ring-die and held the mixture there, so that the center of the mill was practically dry, nor did large lumps of ore accumulate there. Unlike the work of an engine driving stamps, the load on the motor driving Huntington mills varies according to the rate of feed, which must be regulated to the varying hardness of the ore. When a mill is under-fed it races, and by over-feeding it can be brought to rest. Although the feeders were driven

from a pulley on the mill drive-shaft and the speed of rotation of the feeder-plate varied with that of the mill, the Challenge feeder proved to be not sensitive enough to increase or decrease the feed in proportion to the speed of the mill, and it was necessary to detail men to watch the feeders. This difficulty was in part due to the sticky ore.

An objection to the use of Huntington mills is, that they are not covered, and hence, if much water is fed to them, they throw the pulp out at the top, on belts, pulleys, and everything about them. It was the custom to stop the mills for an hour each morning, while the boiler-fires were being cleaned; and at that time the mills were washed clean with hose and broom, to free all accumulated grit from parts where it would work in and cut the metal, and in order that all parts might be inspected and changed where necessary. It paid to give the mill this bath every morning, as the cutting by grit was thereby greatly reduced.

Great Number of Wearing-Parts an Objectionable Feature.

The wear on the mills was great, but would not have been so objectionable but for the number of parts over which it was distributed. The total wear varied from 0.75 to 1 lb. of metal per ton of ore ground, including the wear of such parts as did not show any effect until after six months' or a year's use. There are parts of the mill that show little wear when observed but for a few months, so the actual wear is much greater than is apparent. Some of the large castings have to be scrapped when worn out in certain small areas, while the remainder of the casting is in excellent condition; the scrapping of such castings accounts for the high rate of metal-consumption. The total cost of milling was 60 cents per ton of ore, of which 26 cents was for power (wood used as fuel that cost $3.50 per cord delivered at the boilers), and 9 cents for repair parts. Of the cost of renewal of worn parts, 50 per cent. was for roller-rings, 25 per cent. for ring-dies, 20 per cent. for parts requiring frequent renewal, and 5 per cent. for such parts as were renewed only at long intervals, such as mill-housings, shafts, and gears.

The normal life of the ring-dies was three months; they wore evenly except for a slight flatness just where the feed

entered. It was never necessary to use the grinders that are supplied for truing the dies, and when a die was worn out it was not more than 0.25 in. thick at the thinnest part. It was advisable to replace a ring-die before actual breakage occurred, for if it broke in the mill, the broken ends were apt to spring in and catch the revolving mullers, with disastrous consequences. In Fig. 1 a typical worn die is shown.

The most apparent wear was in the roller-rings, which were made of chrome steel, and had an average life of five weeks. They always wore flat in one or more places before much of the metal had been ground away. In Fig. 1 is shown a worn

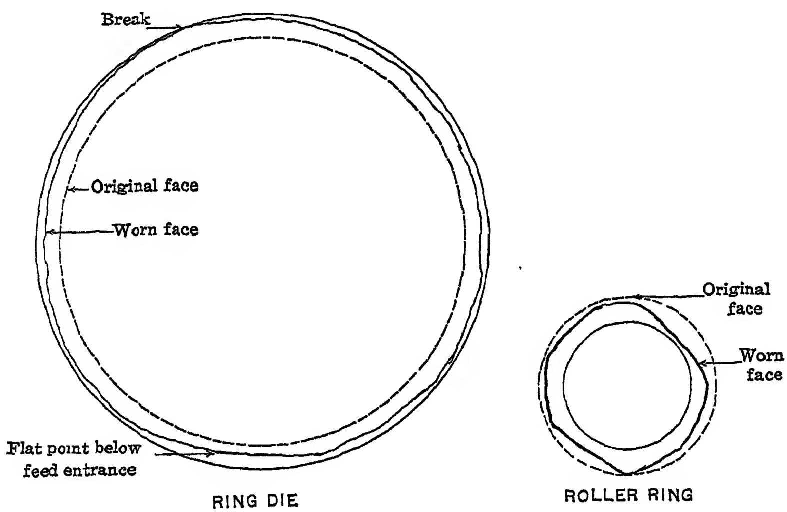

FIG. 1.—DIE AND RING OF HUNTINGTON MILL, SHOWING WEAR.

ring; but it was customary to remove the rings before they had worn out of shape to the extent shown in this illustration. Flat rings caused pounding that, if it had been allowed to continue, would soon have wrecked the mill. The flatness was the result of wearing in one place, due to the ring dragging against the die without turning on the spindle. The rings, when they begin to wear flat, should be removed and trimmed in a lathe. If the flatness is considerable, it will be better to cut away the thicker parts on a planer before turning in the lathe. The quantity of metal cut away by the lathe would be about equal to that from wear in the mill. With a lathe

and planer, the life of a ring can be increased to 10 or 12 weeks, as the ring can be used when very thin, provided it is not flat.

The mullers were inspected each morning, and all flat rings removed. The entire muller with its hanger can readily be withdrawn through the driver; and by having a number of extra heads with the rings wedged on and hangers in place, we were able to change mullers in a few minutes. Ordinarily it is better to use cheaper metal than chrome-steel for the rings, as the time taken to change them is not of much consequence; and for this reason, it is preferable to have the rings take more wear than the dies. As changing a die necessitates removal of the housing from the mill bottom, and it takes 12 hr. to make the change, it is desirable to have the die wear as little as possible compared with the rings. To this end only chrome- or manganese-steel dies should be used.

Bronze Shaft-Bushing Used in Mill-Bottom.

The heaviest part of the mill is the bottom, *A*, Fig. 2, with cone, *B*, cast on; it weighs 7,000 lb. and is an expensive part to replace, especially when such a heavy piece has to be brought to mines difficult of access. The wear is practically confined to the inner bore of the cone where the vertical shaft passes through. If desired, the manufacturers will supply the bottoms with a bronze bushing for the cone, *D*, held in place by set-screws, *E*; this bushing, being softer than the steel of the shaft, takes all the wear, and can be readily replaced when worn, so that the life of the bottom is greatly prolonged.

Housing and Drivers.

The mill-housing slowly wears out, and, after a year's use, holes begin to appear in the back near the feed-hopper, and about the screen-openings. When this first occurred, our mechanics were busy on other repairs; and I had the carpenter repair the housing by cutting blocks of mahogany to fit over the worn places, binding them in place by studs, and by bolts through holes tapped through sound parts of the housing. These blocks were found to wear longer than the iron used for making similar repairs. After that, all such repairs were made with wood, and the blocks did not loosen after a year of service,

except when the bolt-heads wore off. Wooden blocks can be readily shaped to fit the housing and it is much easier to make tight joints with them than with iron. In two years, while all five mills were in use, but one new housing had to be ordered. Gaskets cut from blanketing or canvas are generally used to make a tight joint between housing and bottom; but a cheaper and as serviceable gasket may be obtained by using old hemp rope, fraying out the strands and laying them smoothly on the bottom before lowering the housing to place.

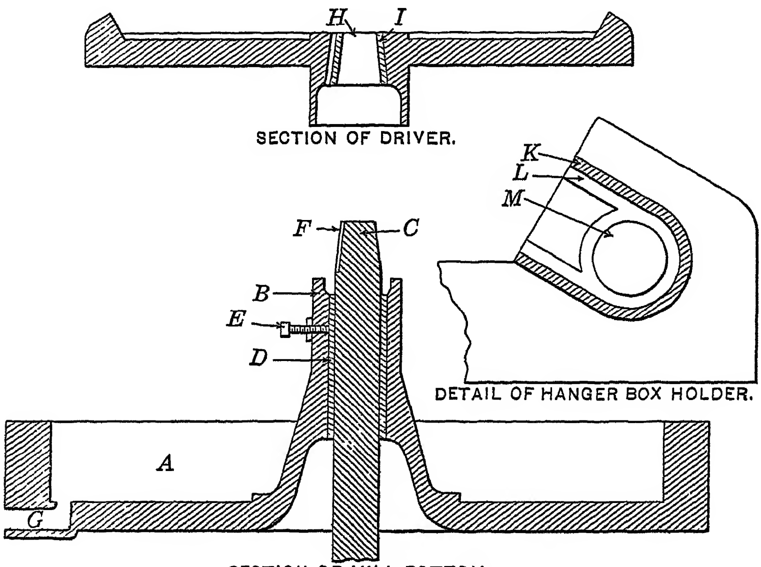

FIG. 2.—DRIVER AND BOTTOM, AND DETAIL OF HANGER-BOX HOLDER OF HUNTINGTON MILL.

The driver or spider carrying the mullers is keyed to the main or central vertical shaft, and rests upon a 0.25-in. shoulder. This shoulder and the bore of the driver wear in time, and trouble ensues through the settling of the driver. This could be remedied by cutting the top of the shaft tapered, without a shoulder, as at *C*, Fig. 2, and cutting the bore, *H*, in the driver, into which the shaft fits, to a corresponding taper, using a key as usual, or preferably a Woodruff key, such as is made for tapered shafts by the Whitney Manufacturing Co. The life of the driver might be prolonged by using a bronze bushing, *I*,

held in place by set-screws, and cut with a key-way slot, so that the key would hold in the driver and shaft, as well as keep the bushing in place.

The drivers wear also in the sockets of the arms that hold the hanger-boxes, *L*. This wear could be taken up by making the casting heavier at the extremities of the arms, so that the sockets could be cut out 0.25 in. deeper, and using a spring-steel U-shaped bushing, *K*, held in place by the spring of the metal.

An Improved Hanger Suggested.

The hangers wear in the arms that are carried in the hanger-boxes; and occasionally the arms break. A sleeve-like bushing could be used over the arms to take the wear, if the recesses in the hanger-boxes were made enough larger to take the bushed arms. A greater improvement would be effected by another design of hanger, with a separately-made steel arm. Such a hanger is suggested in Fig. 3, where a projection, *B*, of the casting, *A*, is cut by a square hole, *C*, through which the steel arm passes, held in place by the key, *D*. This steel arm is square in section where it passes through the hanger; but the ends that are carried by the hanger-boxes, *XX*, are round and cut so as to allow the use of a steel sleeve-bushing, *YY*, over them to take the wear. There would be little possibility of a steel arm breaking. In the figure, *E* is the roller-spindle, held in the hanger by the gib, *F*, and key, *G*, with a set-collar, *J*, for use while adjusting the length of the muller, before the key and gib are brought to bear on the spindle.

The hangers, as now constructed, would not permit the use of a steel arm, since it would cross the hanger where the spindle passes through. The projection, *B*, of the hanger-casting and the hanging of the muller from a steel arm passing through the projection, would result in so shifting the center of gravity that the muller would hang outward if it were not for the ring-die, *N*. But in the usual design of hanger, where the carrying-arm is in the same plane as the axis of the muller, the hanger-boxes are carried far enough out on the driver to cause the ring-die to keep the muller hanging inward as shown in the illustration. This causes the weight of the muller to press against the die; and so the eccentric mounting by a steel

arm, as suggested, would tend to increase still further the pressure of the muller against the die.

In order to have the mullers hang in the proper position, the two-piece, steel-arm hanger would have to be carried by the driver at a point several inches farther from the center of

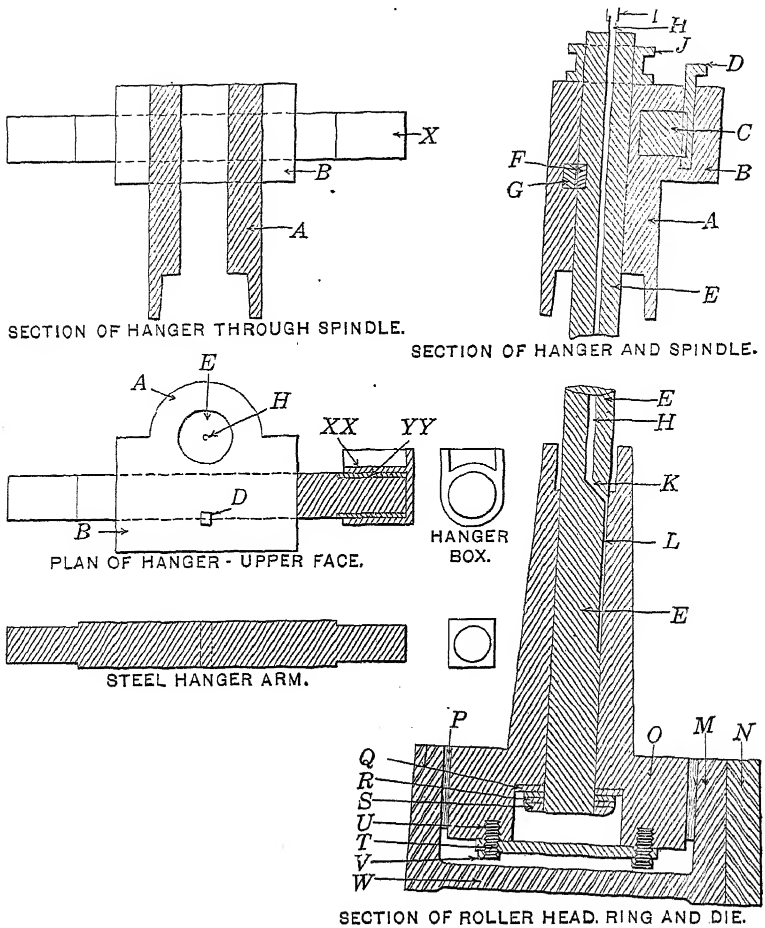

FIG. 3.—HUNTINGTON MILL. DETAILS OF CONSTRUCTION.

the mill than the hanger now in use. This would require a driver of a little greater diameter, and a correspondingly greater diameter for the housing; or the same housing could be retained, and the driver be re-designed of the same diameter, but with the sockets nearer the center of the mill.

The hanger-boxes wear out rapidly, and at an isolated mine

it would be advisable to attempt casting them in the shop. The casting is so simple that it could be easily made by any intelligent mechanic, and a small gasolene melting-furnace could be used to melt the metal in a crucible. If it proved to be difficult to make iron castings in this manner, the boxes could be cast of brass or copper, and the old and worn boxes be melted to make new castings. Molds made of graphite can be obtained for making such castings and are preferable to sand molds when inexperienced mechanics do this work.

Leakage of Lubricant from the Roller-Head.

The most serious wear, with regard to its effect on amalgamation, is in the cover-plate, *T*, of the roller-head, *O*. This plate covers a recess in the head to hold oil or grease for the lubrication of the friction-rings, *R*, and is held in place by four bolts, the heads of which rapidly wear, as also does the cover. The plate comes loose and at times drops to the bottom of the mill—in either of which events, the grease or oil runs out and flows over the amalgamation-plates.

One day I went into the mill while the morning cleaning was going on; and, in washing the sand from the trough around the mill, I noticed some fine gold of green color, concentrated on the rough surface of the casting. When it was washed into an amalgamated pan but little of it was taken up by the mercury, and some of it floated on the water. Treatment with caustic alkalies or cyanide had no effect; but after treatment in an evaporating-dish with strong nitric acid, it assumed a bright yellow color and readily amalgamated. Mineral grease and oil were used in the roller-heads; and it was believed that the oil had coated this gold,preventing amalgamation.

Not only does the cover-plate wear, but the spindle is often cut by grit entering a keyway-like passage or groove cut on one side of the spindle for the purpose of admitting oil to lubricate the bore of the roller-head through which the spindle passes. This groove is usually closed at the top of the spindle by a wooden plug, which, because it is so small, often jars loose, or the mill-men forget to replace it—and then grit works down to cut both spindle and roller-head.

The spindle should be provided with a central passage, as shown at *H*, Fig. 3, passing down the center to a point below

the upper part of the roller-head; the groove *L* should be cut only at that part of the spindle passing through the head, and should be connected with the central passage by a cross passage, *K*; the upper part of the central passage should be threaded to take a small closing-plug, *I*, that can be screwed in tightly with a wrench. A little oil should be added each morning when the mills are being cleaned; and the spindles should be washed free of all dirt before removing the plug.

The leakage of oil may be prevented by designing the roller-head and roller-ring as shown in Fig. 3. The lower part of the ring is closed by forging the metal so that it is open at the top and closed at the bottom, as shown at *W*. The roller-head is of about the usual design, but the bolts to keep the cover-plate, *T*, on the oil-chamber are replaced by studs, *V*. There are then no bolt-heads to be worn off at the top of the head, and the diaphragm-like closing of the roller-ring prevents wear of the cover-plate. No oil can possibly flow into the mill. These roller-rings would be more expensive and more difficult to make, but the advantages of such a construction would fully justify the greater cost.

Wedges, Screens, and Belts.

Wooden wedges were used to hold the ring-die in place in the mill-bottom and for binding the roller-rings to the heads, as shown at *P*. The wedges were driven close together and spread by thin steel wedges. In the tropics, *leche amarilla* is the best material obtainable for wedges, but mahogany or pine may be used, though they are not so good. The wooden wedges are perfectly satisfactory for the purpose of binding rings and dies.

Brass wire screens have an average life of three days; needle-punched round hole, 20-gauge iron screens, with apertures equal to 25-mesh, wear from 7 to 10 days. The screens in Huntington mills are subjected to hard blows from the larger pieces of ore being thrown violently against them by the mullers. The punched iron screens are preferable on this account; but they reduce the capacity of the mill to a slight but appreciable extent.

The mill should be equipped with covers or guards extending above the housing, to prevent pulp being thrown on the belts. Grit soon wears the belts, the faces of the pulleys, and eventu-

ally works down into the bevel-gears, causing them to wear rapidly. Apart from the wear of ring-dies and roller-shells, it is true that about 90 per cent. of the wear on the Huntington mill is due to grit alone. The driving-shafts should be so placed that the belts run at flat angles, preferably not over 45°, since vertical belts slip if not tight, and the evils of tight belts are more apparent in running this class of machine than perhaps in any other.

Conclusions.

Huntington mills are well adapted for grinding soft clayey ore where only the lightest stamp could be used, if stamps could be used at all with any degree of satisfaction—and then only at low efficiency; they discharge the pulp, as soon as it is ground fine enough to pass the screen, more rapidly and more completely than do any other forms of grinding-machines, and therefore little pulp ground fine enough to pass the screen remains in the mill to be slimed. It is this high screening-efficiency that makes the machine so adaptable for the regrinding of tailings before concentration; but because of the wear and number of its wearing-parts, other forms of machine are used where the Huntington mill would give a product with much less slime. In the concentrator of the Nevada Consolidated, both Chile and Huntington mills are used for regrinding, but the repair-costs for the last are much greater. The Tonopah Mining Co. intends to replace with Chile mills the Huntington mills in use in its mill, solely on account of the greater wear of the Huntington.

This excessive wear may be reduced by such changes as have been suggested above, the idea being to reduce the wear to small parts that can be readily replaced at little cost. The objection to the mill is not so much the actual wear, but that certain parts of the mill wear out in one spot, so that a large casting, little worn in other places, has to be scrapped. The power required is not great, and compares favorably with other grinding-machines; the low cost, simplicity, and small expense of foundation and installation, are real advantages, which are unfortunately outweighed, for the Huntington mill as at present constructed, by more important considerations above mentioned—except in cases where no other mill is so perfectly suited to meet the conditions imposed by the nature of the ore.

Canadian Mining-Law.

BY J. M. CLARK, LL.B., K.C., TORONTO, CANADA.*

(Wilkes-Barre Meeting, June, 1911.)

For some years past, those interested in the development of the increasingly important mining industry of Canada, have urged the adoption by the Dominion Parliament of a federal mining-law, which would have the force and stability of statutory enactment. At present, placer-mining in the Yukon Territory is governed by the Yukon Placer Mining Act. All other mining under federal jurisdiction is governed by Orders in Council and Ministerial Regulations.

In the earlier stages of development, it is perhaps a matter of necessity that these important matters should be so dealt with; but it is now felt that the time has come when mining-rights in the extensive regions under federal control should be put on a permanent basis, and that any changes required from time to time should be made only after full and open discussion in Parliament.

A short sketch will suffice to indicate how vast and varied the interests affected really are.

When the Dominion of Canada was constituted by the Imperial Statute known as the British North America Act of 1867 (which came into force by proclamation on July 1 of that year), it comprised only the present Provinces of Ontario, Quebec, Nova Scotia, and New Brunswick; but provision was made for the inclusion of Newfoundland, Prince Edward Island, British Columbia, Rupert's Land, and the North West Territories. Subsequently Rupert's Land and the North West Territories were acquired, the Crown Colonies of British Columbia and

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* Secretary's Note.—Mr. Clark, an eminent Canadian lawyer, and joint author of the treatise on "The Law of Mines in Canada," has been requested by the Dominion Government to prepare a Federal mining-law, and presents this paper, by invitation of the Council, in order to obtain, if possible, useful suggestions from members of the Institute.—R. W R.

Prince Edward Island were admitted, and all the other British Territories and possessions in North America, with the islands adjacent thereto, except Newfoundland and its dependencies, were annexed to Canada by Great Britain.

Canada, consequently, now comprises the whole of the northern half of North America, except Alaska, Newfoundland, and that portion of Labrador which constitutes a dependency of Newfoundland. All lands, mines, minerals, and royalties belonging at the time of the union to the several Provinces of Canada (now Ontario and Quebec), Nova Scotia, and New Brunswick, are declared to belong to that one of the said several Provinces of Ontario, Quebec, Nova Scotia, and New Brunswick, in which the same are situated or have their legal origin—subject, however, to any trusts existing in respect thereof, or any interest therein, other than that of the Province.

Each of the Provinces named has jurisdiction to make laws for the management and sale of its public lands, and of the timber-wood thereon, and also as to property and civil rights in the Province.

With some exceptions, not necessary to be here specified, the same rules were made applicable to Prince Edward Island and British Columbia. But very different conditions and regulations obtain in the remaining parts of Canada.

Under the sanction of an Imperial Statute, the Dominion of Canada obtained a surrender of the lands and territories granted by Charles II. in 1670 to the Governor and Company of Adventurers Trading into Hudson Bay, known as the Hudson Bay Co.; and Rupert's Land and the North West Territory were consequently admitted into the Dominion as of July 15, 1870.

When the Provinces of Manitoba, Saskatchewan, and Alberta were formed, the lands, mines, and minerals, with slight exceptions, were not transferred to the Provinces, but remained the property of the Dominion of Canada, and subject to federal jurisdiction and control.

The proposed federal mining-law must deal with the mines and minerals of these three Provinces, of all the Territories (including the Yukon Territory), and of certain areas of the older Provinces, principally the Indian lands and the Railway belts of British Columbia. It must, therefore, deal with placer-

mining, coal, natural gas, oil, petroleum, gold, silver, copper, and the other minerals. The whole field must be covered and every problem of mining-law solved.

The framing of this general law is regarded by mining-men as supremely important, not only on account of the great interests actually and potentially involved, but also because it is looked upon as the first step towards the unification of the mining-laws of Canada. The vital importance of such completeness, wisdom, and practical convenience being presented by the federal statute as will recommend it to the several Provinces for voluntary adoption is therefore self-evident.

While the Dominion has no jurisdiction over the mining-laws of the Provinces which own mining-lands, it is hoped that the provisions of the federal law, by reason of their excellence and efficiency, will gradually be adopted by the various Provinces.

To paraphrase a famous saying, this must take place, not by reason of imperial power, but by the imperial power of reason.

In this connection, a striking instance of concerted action by independent jurisdictions may be mentioned. Some years ago, an exceedingly well-drawn Act, dealing with bills of exchange and promissory notes, was passed by the Imperial Parliament. The same Act, with slight changes, was passed by the Canadian Parliament, and by a majority of the State legislatures of the United States; so that it may now be said that this statute governs the greater part of the English-speaking world!

There is no reason why the members of this Institute should not take a useful and active part in obtaining for the mining world advantages similar to those which have been thus secured by the mercantile communities of Great Britain, the United States, and Canada.

At the present time, a discussion of the fundamental principles upon which such a mining-law as is proposed should be based, and of the merits and deficiencies of such codes as that of Mexico, would be interesting and instructive, as bringing together, in useful form, the results of close observation and varied experience of the mining-laws of the world.

There is no danger that any form of the so-called "apex-law" will be again introduced into Canada. That law was once copied, under the influence of miners from the Pacific

States, by British Columbia, but was finally abolished April 23, 1892, since which date the rights of the holder of a mineral claim are confined, in British Columbia, as in all other parts of Canada, to the ground bounded by vertical planes drawn through its surface boundary-lines. The vested rights of claim-owners who had located their claims under former acts were protected; and the "apex-law," in British Columbia, as elsewhere, has given rise to costly litigation, which seems inherent in the system of extra-lateral rights.

There are, however, other important questions to be discussed: such as how adequately to protect the prospector, without at the same time introducing the danger of "blanketing;" the function of discovery in the acquisition of mining-title; the most useful forms of working-conditions, and the most efficient methods of enforcing such regulations. Last, but not least, the ever-present and ever-troublesome questions of taxation and royalties must be considered.

Discussion.

Rossiter W. Raymond, New York, N. Y.:—It is satisfactory, but not surprising, to learn that there is no danger of the adoption in Canada of the apex-law with its extra-lateral right. I do not think that any community which has once experienced the evils of that system, and has escaped from them by abandoning it, would ever dream of returning to it. And British Columbia having had that experience, has doubtless furnished a sufficient object-lesson for the whole Dominion.

Mr. Clark's hope that a federal law may be framed which will ultimately be adopted by the Provinces, is not chimerical. Not only the commercial instance which he cites, but the history of our United States law, encourages such a hope. That law prescribes a few conditions, leaving to local legislation freedom to ordain others, not inconsistent therewith. For instance, the form and the maximum dimensions of a mining-claim and the minimum amount of annual "assessment-work," are prescribed, together with a few forms of procedure; but smaller dimensions, larger amounts of annual work by possessory owners, and additional forms of procedure, may be imposed by local legislation or regulation. In many cases,

especially during the earlier period following the adoption of the federal law (1870), this freedom was abundantly used, and locators of lodes or placers were often obliged to do, for the maintenance of their possessory titles, a good deal more than the U. S. statutes required; but gradually the convenience of uniform conditions, working quietly, but continuously, like the pressure of gravity, had its effect; and, at the present time, the local regulations of mining-districts have been largely superseded by State or Territorial statutes, which are, in the main, not only consistent, but identical with those of the federal law. Another illustration is the manner of making and recording a mining-location upon the public domain. It may seem strange that the U. S. law prescribes nothing at all in this respect. It required the location to have certain essential features of shape, maximum dimensions, and relations to the discovery of a mineral deposit within its boundaries; but it does not require any particular form of record or proof of these fundamental requirements. Indeed, the United States government does not to-day possess either records or maps showing what portions of its public mineral lands have been appropriated by valid mining-locations, and, being held under possessory title, do not now belong to that domain. The explanation of this anomaly is historical. At the time when our government had to do something in order to define its relations with miners who were technical trespassers upon the public lands, those lands constituted, in the main, a vast unsurveyed wilderness. A theoretically perfect and properly guarded system for the record of mining-titles would have been impracticable of execution; and Congress, therefore, did the best it could under the circumstances. The punishment of trespassers being neither desirable nor practicable, it legalized the trespass, and left the parties concerned to settle their relations to one another according to the local rules which they had themselves adopted in the several mining-districts, or which might be established for them by local legislatures, subject to a few more or less elastic requirements established by the United States, as the real owner of the land. Meanwhile, the facts concerned were left to be proved by any kind of evidence, documentary or oral.

In my judgment, this action of Congress, though warranted under the circumstances so far as records of location, etc.,

were concerned, might and could have been remedied, when these circumstances had greatly changed, by requiring records of location, etc., to be made in or officially transmitted to the U. S. local or General land-office. But it is worthy of note that, without any such requirement, the effect of simple considerations of the certainty and safety of such records has brought about a general uniformity of local legislation, requiring them to be filed with the officers of courts or counties, who will be responsible for their preservation from mutilation or destruction. It is not yet the duty of such officials to give notice to the United States of such entries, affecting the title of the United States to its public lands; but that step may easily be taken. Meanwhile, this narrative of somewhat chaotic progress may encourage the belief that obviously wise and useful features of administration will, in the end, be adopted by communities upon which, when first promulgated, they are not legally binding. In other words, it is worth while for a federal government, like that of the U. S. or the Dominion of Canada, to frame a system of mining-law for its own lands, which will commend the acceptance of its constituent States or Provinces in the administration of their own lands.

To this end, I think the first requisite is a survey of such lands. Apart from the mischievous extra-lateral right, the greatest cause of confusion and waste in those mining-districts of this country which have been afflicted by our mineral-land law, has been the lack of such public surveys as would permit the accurate definition of a mining-location by reference to established landmarks. I do not know how far the Dominion has proceeded in the discharge of this public duty—one of the very first, in my opinion, which is incumbent upon any government worthy of the name. At all events, I hope that Mr. Clark's draft of a code will include provision for immediate performance of this work. Mining-grants may have to be made in territory not yet surveyed; but this should be done under conditions which will secure their subsequent re-definition by reference to the lines of such a survey, and will permit the readjustment of their boundaries so as to conform, if possible, to those lines. This will be comparatively easy, if the boundaries of the original location be required to follow the direction of the future survey-lines—*e.g.*, to run N–S. and E–W. The

purchaser or possessory tenant of a tract of mining-land would never object to paying for a little more area here, or a little less there, in order to conform to this obviously convenient rule—provided, of course, he were not haunted by the fear of losing problematical "apex-rights" by any variation in his lines. Under our present U. S. system, the locator determines, as well as he can, the course of the "apex," which he fondly hopes he has truly discovered, and is bound to claim a rectangle covering that course—under penalty of losing some or all rights, both extra-lateral and intra-lateral, if later developments should prove him mistaken. It might be a hardship to him to be required to draw his boundary-lines in particular directions; although I am inclined to think that, in the long run and in the majority of cases, the result would be advantageous to our mining-operators, by reason of their greater security of title. I have had to do with a large number of mining-litigations, and I can recall few lode-claims involved in such cases from which the lode did not depart, at some point, across a side-line; so that I am inclined to believe that, even under the "apex-law," the boundaries of locations might have been required (with some modification or conditions as to length and width), to run N–S. and E–W., with real advantage to locator. Be this as it may, I see nothing to prevent the adoption by the Dominion of Canada of a provision so well-established and so universally approved in the sale of public agricultural lands.

Next in importance to public surveys is the official classification of the public lands to be leased, sold, or opened to prospectors. This classification should be, in my opinion, final and conclusive. If the land in question has been sold outright, say, at the price of agricultural land, and the grant or patent of the government, conveying the full common-law fee simple to its contents, *usque ad astra*, *usque ad inferos*, has been issued to the purchaser, then the original official classification of it, as agricultural, should not be disturbed by proceedings attacking the purchaser's title, on the ground that it was or is, in fact, more valuable for mining than for agricultural purposes. The government should occupy, in this respect, precisely the position of a private seller. (Of course, actual fraud, to which the purchaser was a party, might be pleaded against his title, but

to no further extent and under no other conditions by the government than by any private party wronged by such fraud. That is to say, the government should itself bring suit for the abrogation of its grant or deed; and the latter should not be open to collateral attack in any private suit.) In short, the purchaser of anything from the government is entitled, in justice as well as policy, to know just what he gets, and to be assured that he really gets it. The danger that, through an incorrect official classification, mineral land may be sold at a lower price as agricultural land, is entirely insignificant compared with the importance of giving a clear and secure title to purchasers.

On the other hand, lands may be granted for agricultural purposes, with a reservation by the government of the mineral rights. In this case, a previous official classification is less important. Yet I think it might well be required to protect the government against unnecessary administrative complications. Any land which is officially classed as "mineral," had better not be sold as "agricultural;" and, in any case, it is best that in such transactions, as in private bargains, both parties should clearly know what they are doing. In leases of mineral rights, it might be urged that the government should be able to increase its requirements upon proof of unexpected value of the property. One obvious answer is, that such a change should be practicable, if at all, only after a term of years. But a more conclusive answer is, that the mining industry should be taxed upon its annual product or profits; and such a tax will take care of all unexpected prosperity, without disturbing the conditions of mining-title. I feel bound to say, however, that nearly 50 years of observation and experience have inclined me to believe that the acquisition by private parties or corporations of the full fee simple of public lands, including the mineral right, is better in the long run, than any system of leasing by the government. If such a system should be deemed advisable, then the condition of the retention of title should be, not a given amount of annual "work," but an annual payment of money. The requirement of "assessment-work," under our U. S. law, is delusive and useless. The required annual payment of a sum of money would be much more effective in preventing the

retention of possessory titles (which are practically, under our law, government leases) for speculative purposes.

The policy of requiring continued work as a condition of continued possessory tenure is not particularly harmful with regard to metal-mines, especially when, as with us, the amount of such work is trivial; but the governmental leasing of coal-lands for limited periods, and upon conditions of continual operation under penalty of forfeiture, is thoroughly bad. This idea has been suggested, I believe, by President Taft himself, whose sane and wise views of the general subject of "conservation" have won the approval of intelligent people. But I think he is wrong on this point. The operation of a colliery by a lessee is certain to cause the sacrifice of future to present interests; and the requirement that such a lessee shall keep going or lose his lease simply aggravates the situation. No governmental inspector could fairly require of a lessee the expenditure of a large sum of money which he might never recover, under an arrangement subject at any time to executive cancellation, especially if such expenditure were required, not for the safety of workmen, but only for the advantage of some future lessee; and the requirement of continuous operation as a condition of tenancy, would operate to favor that over-production which is the greatest enemy of "conservation." At a recent meeting of the Institute, an eminent authority on this subject[1] read a paper advocating the shutting-down by the government of coal-mines that did not pay, in order to prevent the injurious over-production and consequent waste of coal. My proposition is, that such mines will be shut down by their proprietors without governmental interference, provided they are not forced to continue operations for some other reason, such as the danger of thereby losing their property altogether. In short, I think that, in all such questions, private ownership and liberty are likely to produce better results than governmental supervision; and that the best thing any government can do is to preserve order, enforce contracts, give to lessees or purchasers of its lands clear and valid titles, and then allow them the largest practicable liberty of enterprise and industry—

[1] Edward W. Parker, The Conservation of Coal in the United States, *Trans.*, xl., 601 (1910).

reaping its own advantages, not from the extortion of a percentage of the anticipated results of speculative adventures, but from the consequent increased wealth of all its people and the fair taxation of that wealth.

I could say many other things upon the text which Mr. Clark has presented, but I trust the foregoing will incite other members of the Institute to offer suggestions which may be useful in his undertaking.

A Drafting-Table for Tracing Through Opaque Paper.

BY A. T. SCHWENNESEN, STANFORD UNIVERSITY, CAL.

(Wilkes-Barre Meeting, June, 1911.)

EVERY engineer has occasion to trace or copy a map, plan, or other drawing on paper too thick for the ordinary way of using tracing-cloth or tracing-paper. When the figure is small and simple a copy may be made by holding the original against a window-pane, covering it with the paper, and tracing direct by the aid of the strong sunlight from outside. The need of utilizing this principle on a larger scale and in a more convenient position led Dr. J. C. Branner to plan the table of which the following is a description:

This table was first made in the form of an adjustable glass-top table with a mirror beneath, in 1887, while Dr. Branner was State Geologist of Arkansas. Later it was modified as experience suggested until the form as here described was evolved.

The device consists essentially of a drafting-table with a plate-glass top, upon which the original drawing and the paper are laid, and a mirror mounted underneath to reflect the light of the sky up through the drawing. The glass top is hinged and fitted with two arms and thumb-screws, so that it can be raised and fixed to any position, either inclined or horizontal. The mirror is pivoted and revolves about a horizontal axis, so that it may be tilted to any angle. The hood of cardboard or black cloth prevents the reflection of light from the tracing, and may or may not be attached to the table.

The apparatus is set up before a window through which part of the unobstructed sky is visible. The mirror is then adjusted like the reflector of a microscope, so that the sky light is reflected up through the drawing. If the mirror can be so located that the direct rays of the sun are reflected through the drawings, thicker paper can be used.

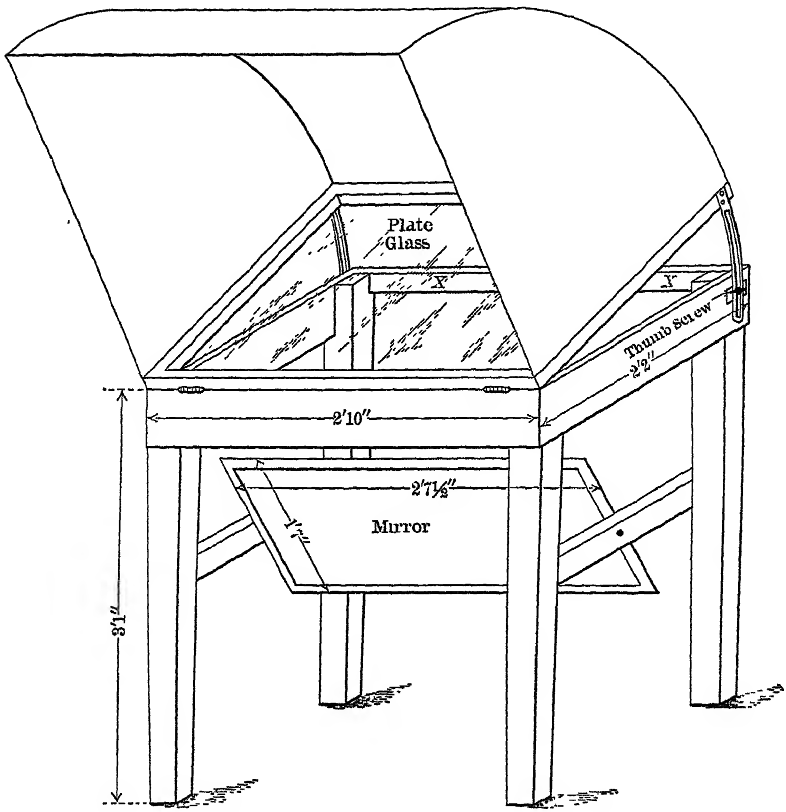

FIG. 1.—ELEVATION OF GLASS-TOP DRAFTING-TABLE, WITH HOOD.

The map or drawing may be held in place by clips screwed to the top of the plate-glass frame or by lead weights placed on top of it.

Fig. 1 gives the dimensions and shows the general appearance of the table in use in the department of Geology and Mining at Stanford University. The dimensions may be varied to suit individual needs. An important point to be remem-

bered in the construction is that the piece marked *X* should be made as narrow as possible so as not to shut out more light than necessary. The frame of the glass top also should be made narrow at the top, for the same reason.

This table can be used at night by employing an electric light, so placed as to be reflected or even to shine directly up through the plate-glass table-top.

It sometimes happens that the light from beneath is inconveniently strong, but this objection can be obviated by cutting a small opening in a piece of thick or dark paper which is laid over the drawing. The tracing can then be done through the hole, and the sheet can be moved about at pleasure, which gives the advantage also of preventing the tracing from being soiled, and it often brings out more clearly the lines to be traced.

The Universal Metalloscope—A Perfected Microscope for the Examination of Metals.

BY ALBERT SAUVEUR,* CAMBRIDGE, MASS.

(Wilkes-Barre Meeting, June, 1911.)

THE instrument about to be described meets so perfectly the special needs of the metal microscopist that there seems to be little doubt but its merits must be readily appreciated by those who have had any experience in the microscopical examination of metals.

The Microscope-Stand.—The microscope-stand proper, Fig. 1, consists of a microscope-tube, provided with both coarse and fine adjustments of the best construction, and with a draw-tube, rigidly mounted on a bar supported at both ends on substantial and firm cast-iron legs. The height between the table and the under side of the supporting bar is 5 in. and the distance between the supporting legs 12 inches.

This arrangement affords free space below the objective for the examination of large specimens of metals, such as full rail-sections, without detracting in the least from the value of the

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* Professor of Metallurgy and Metallography in Harvard University, Cambridge, Mass.

instrument when applied to the examination of the usual small specimen, as explained later. Many metal microscopists frequently have to examine bulky specimens, and this is altogether impossible with the ordinary microscopes as well as with the

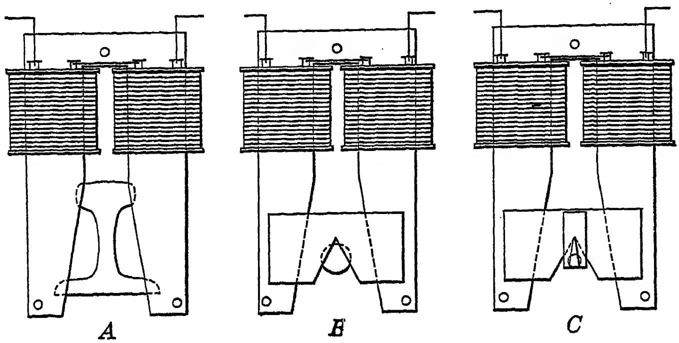


FIG. 2.—(*A*) ELECTRO-MAGNETIC STAGE AND RAIL SECTION. (*B*) ELECTRO-MAGNETIC STAGE, TEMPLET, AND MEDIUM-SIZE SPECIMEN. (*C*) ELECTRO-MAGNETIC STAGE, TWO TEMPLETS, AND SMALL SPECIMEN.

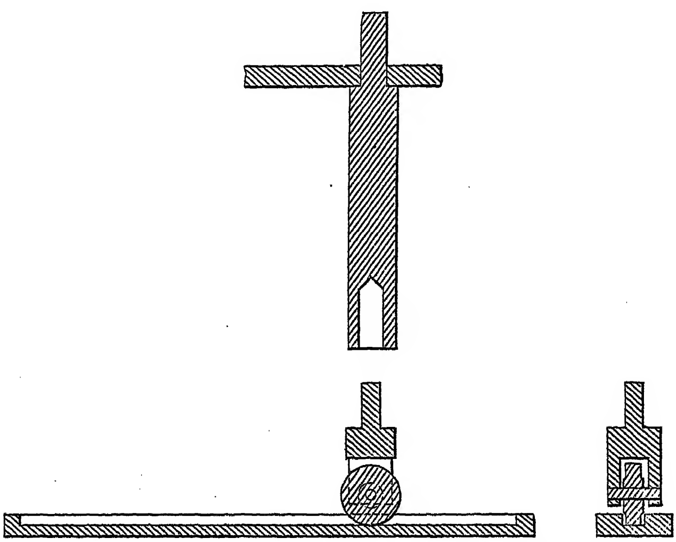


FIG. 3.—BACK LEG OF ELECTRO-MAGNETIC STAGE AND SLIDING-PLATE.

special metallurgical microscopes which have been designed and described from time to time.

Recourse must be had to all sorts of makeshifts for the proper support of large specimens, or, more often, the microscopist

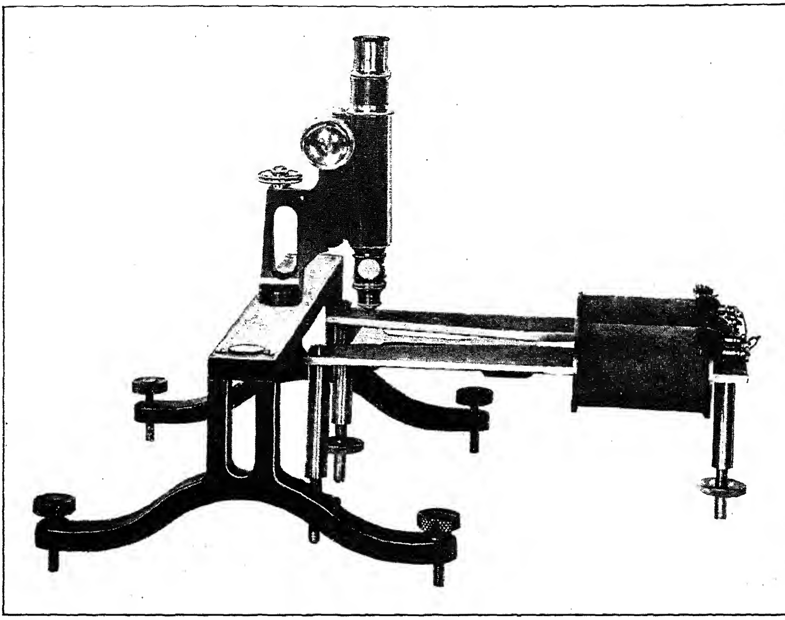


FIG. 1.—UNIVERSAL METALLOSCOPE: STAND, EYE-PIECE, VERTICAL ILLUMINATOR, OBJECTIVE, ELECTRO-MAGNETIC STAGE, AND RAIL SECTION.

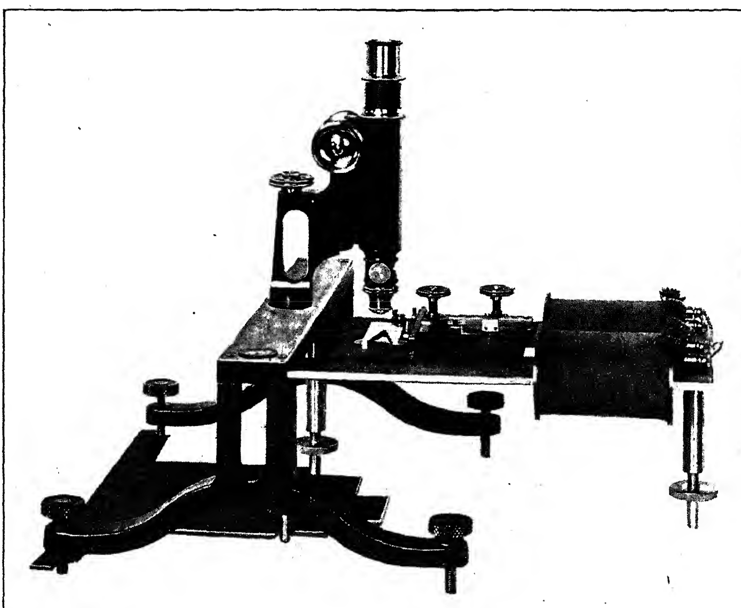


FIG. 4.—UNIVERSAL METALLOSCOPE: ELECTRO-MAGNETIC STAGE WITH MECHANICAL STAGE, MAGNETIC SPECIMEN-HOLDER, SMALL SPECIMEN, AND BASE-PLATE.

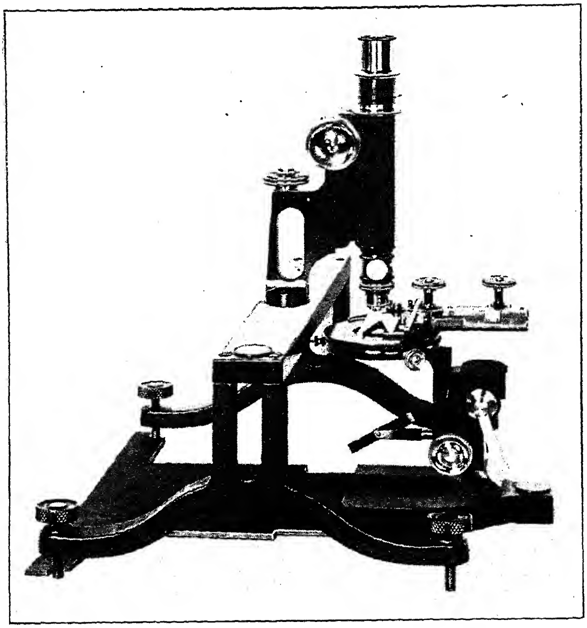


FIG. 5.—UNIVERSAL METALLOSCOPE: MECHANICAL STAGE ON HORSESHOE BASE, MAGNETIC SPECIMEN-HOLDER, SMALL SPECIMEN, AND BASE-PLATE.

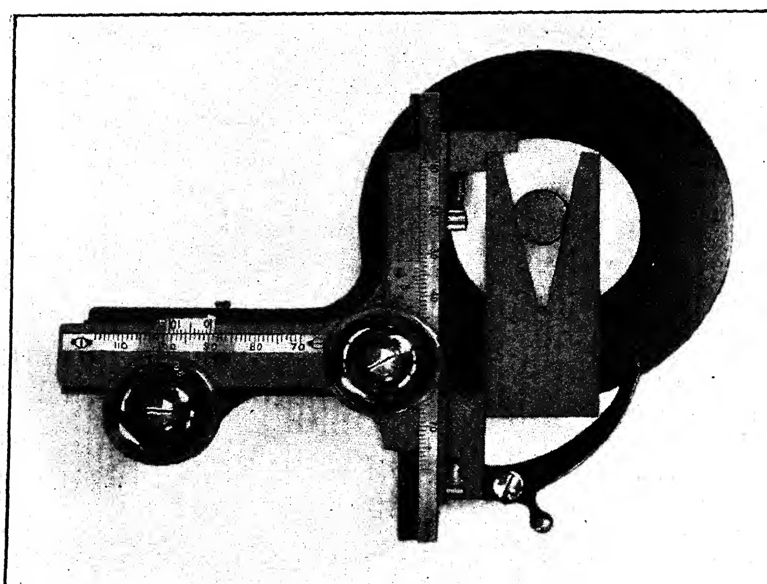
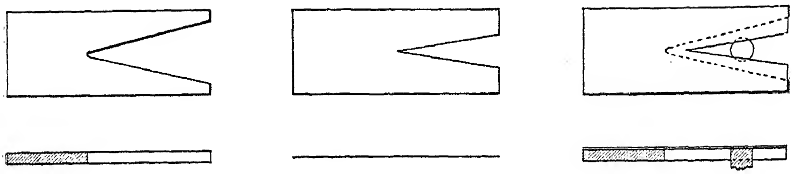


FIG. 6.—MECHANICAL STAGE AND MAGNETIC SPECIMEN-HOLDER.



A *B* *C*

FIG. 7.—(*A*) MAGNETIC HOLDER. (*B*) STEEL TEMPLET. (*C*) MAGNETIC HOLDER, TEMPLET, AND SAMPLE.

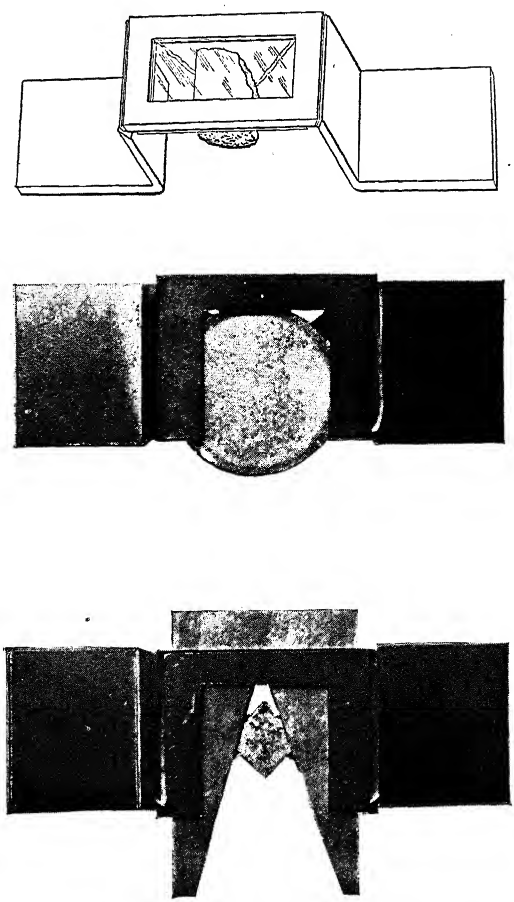


FIG. 8.—SPECIMEN-HOLDERS FOR NON-MAGNETIC SPECIMENS.

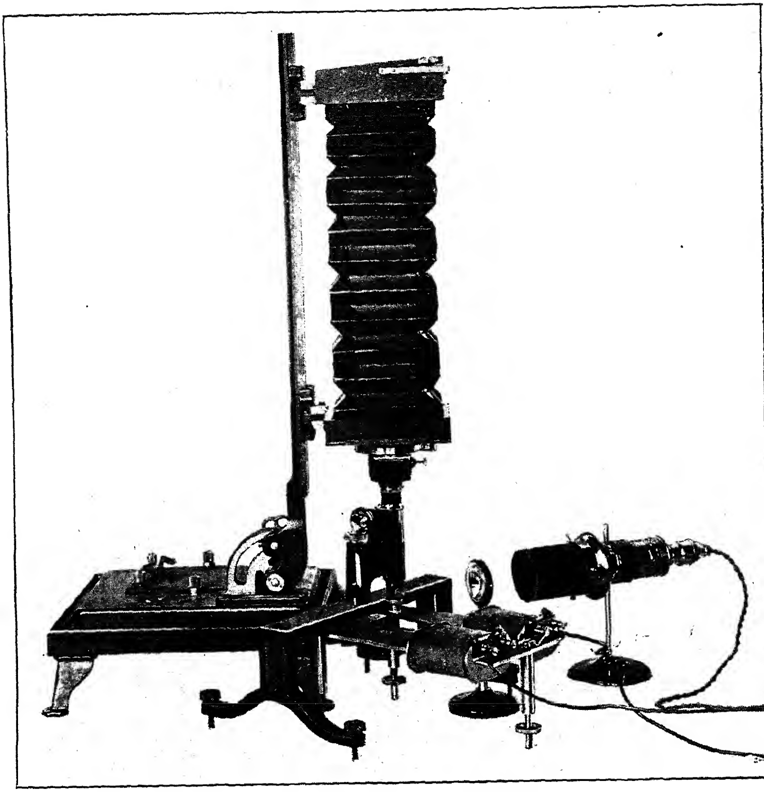


FIG. 9.—UNIVERSAL METALLOSCOPE, NERNST LAMP OUTFIT, AND VERTICAL CAMERA.

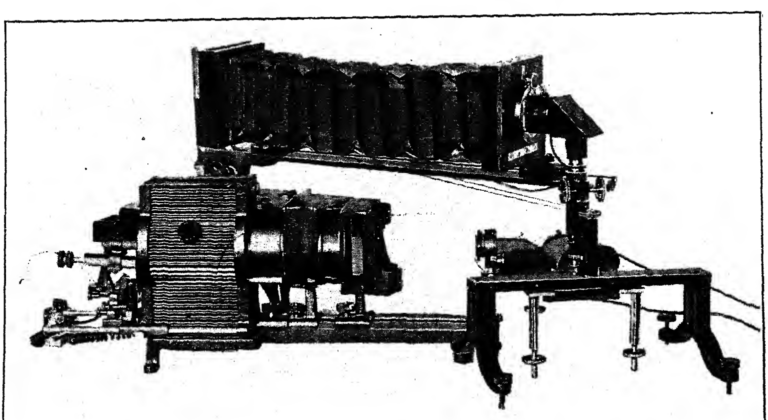


FIG. 10.—UNIVERSAL METALLOSCOPE, ARC LAMP OUTFIT, LARGE TOTALLY-

gives up the attempt altogether, or else resigns himself to the cutting of the bulky samples into small pieces to be laboriously polished and separately examined.

It is believed that an instrument permitting the examination of large as well as of small specimens with equal ease and accuracy will be welcomed by metallographists, and that it will lead to more frequent examinations of full sections of metal implements, a departure which should bring fruitful results.

Electro-Magnetic Stage.—The perplexing question of the proper support, for microscopical examination, of iron and steel specimens of all sizes and shapes has been most happily and effectively solved by the use of the electro-magnetic stage illustrated in Fig. 1. This stage consists of a steel plate 7 by 14 in. having a V-shaped opening, and converted into a powerful electromagnet by means of two bobbins with solenoids surrounding the arms of the steel plate, as clearly shown in the illustration. Electrical connection is readily made with any suitable current, and the use of an incandescent lamp in series provides in a simple way the necessary outside resistance to prevent heating of the solenoids. Large specimens of iron and steel, such as rail-sections, *A*, Fig. 2, are firmly held in an accurate position by the attraction of the magnetic stage, the extremities of the flange only and a narrow space on each side of the head being hidden from view. The size and shape of the stage-opening make possible the ready support of specimens measuring from 2 to 6 in. in greatest dimension.

Templets for the Examination of Small Specimens.—For the examination of iron and steel samples from 2 in. in length down to the smallest dimensions, a steel templet, also with a V-shaped opening, is placed on the stage, shown at *B*, Fig. 2. This templet through its contact with the stage becomes strongly magnetized and the specimens to be examined are suspended to it.

For the examination of very small specimens with high-power lenses the thickness of this templet would prevent the necessary close approach of the objective. To make this approach possible a very thin steel templet (not exceeding 0.01 in. thick) is used, shown at *C*, Fig. 2, which makes possible actual contact between a high-power objective and the smallest specimen.

Support of Non-Magnetic Specimens.—For the support of non-magnetic specimens, such as non-ferrous metals, rocks, cement, etc., a very simple device is provided, consisting of two cross-bars and rubber bands, which is readily attached to the stage and by means of which the non-magnetic specimens, as well as the templets when needed, are firmly held in place regardless of their size or shape.

Leveling-Devices of Stand and Stage.—It is, of course, essential, especially when using high-power objectives, that the optical axis of the microscope be accurately perpendicular to the surface under examination. To secure this result both the stand and the stage are provided with leveling-screws, as shown in Fig. 1. For leveling the stage a small spirit-level may be placed upon it, or better, upon the sample under examination, and the necessary adjustment quickly made. For leveling the microscope-stand the eye-piece should be removed, the small level placed on top of the tube and the leveling-screws adjusted. By placing the instruments on a table or desk having a smooth and flat top, it is evident that, barring accidents, the stand and stage will remain indefinitely accurately leveled.

Motion of the Stage.—In order to examine the entire surface of a large specimen it is necessary to bring in turn within the field of the microscope the different portions of the specimen, and this necessitates the moving of the stage in various directions. The weight of the stage, however, would create considerable friction between the legs and the supporting table, making the sliding-motion jerky and otherwise unsteady. To overcome this difficulty the back leg of the stage is provided with a small wheel running in a groove cut in a small brass plate fastened to the table or desk, shown in Fig. 3. The mounting of the wheel is provided with a pivot fitting snugly into a hole in the leg. This construction makes possible the ready back-and-forth motion of the stage, as well as its free circular displacement around the axis of the back leg, thus permitting to bring quickly any desired portion of the object under the objective. As the bulk of the weight is supported by the back leg, the arrangement makes possible a very steady and smooth motion of the stage.

Mechanical Stage.—The use of a mechanical stage is often highly desirable. This is taken care of in the present instru-

ment in two different ways: (1) by the use of a mechanical stage suitably attached to the electro-magnetic stage, and (2) by the use of a mechanical stage independently mounted on a separate base of the usual horseshoe pattern.

The first method is illustrated in Fig. 4. A mechanical stage of usual construction is screwed on a brass plate provided with two small pins fitting two corresponding holes in the magnetic stage, thus securing a firm and constant position for the mechanical stage. When using a mechanical stage, however, a rigid and constant position should also be secured between it and the microscope-stand. To that effect a brass plate is provided, with recesses to receive the back legs of the stand as well as the front legs of the stage, shown in Fig. 4. It is then possible at any time to place the microscope-stand and the stage in exactly the same relative positions.

The second method consists in the use of a mechanical stage separately mounted on an ordinary horseshoe base, shown in Fig. 5. To secure a constant relative position between stand and stage, the foot of the latter fits into recesses provided for that purpose in the base-plate.

The use of this independently mounted mechanical stage offers the additional advantage resulting from the vertical up-and-down racking of the stage, rendering unnecessary any vertical adjustment of the light and condenser, as well understood by metallographists.

Specimen-Holder for Mechanical Stage.—When using a mechanical stage the electro-magnetism of the large stage cannot be utilized for suspending the specimens. In this case, however, the specimens are necessarily small, and the small permanent steel magnet illustrated in Figs. 6 and 7 is in every way satisfactory. The central opening of the stage should not be less than 1⅝ in. in diameter, as this will permit the ready suspension of the specimens as well as their removal from the stage, making it unnecessary ever to remove the magnetic specimen-holder, which is clipped to the stage like any ordinary slide. A thin templet is provided, as shown in Fig. 7, for the examination of very small specimens by high-power objectives when the front lens of the objective must approach the object very closely indeed.

In case of non-magnetic specimens, the holders shown in

Fig. 8, and which have been so widely used for many years, will be found most satisfactory.

Examination of Transparent Objects.—To adapt the universal metalloscope to the examination of transparent objects, thereby converting it into an ordinary microscope, or, if desired, into a petrographical microscope, a separate stage on horseshoe base should be used, as shown in Fig. 5, when the necessary Abbe condenser, analyzer, polarizer, etc., can readily be attached. The instrument is then in no way inferior to high-class microscopes for examination by transmitted oı polarized light.

Illumination.—Artificial illumination is universally used for the examination of metals, the sources of light which have been found most satisfactory being, in the order of their excellence, intensity, and decreasing cost: (1) the electric-arc lamp, (2) the Nernst lamp, and (3) the Welsbach gas-lamp.

The Welsbach lamp outfit is very inexpensive and quite satisfactory for visual examination by low- and medium high-power objectives. In taking photo-micrographs, however, its lack of intensity necessitates very long exposures, while with high-power objectives the light received upon the camera-screen is so faint as to render proper focusing of the object a very difficult, if not impossible, operation. The electric-arc lamp provides by far the best means of illuminating metallic samples. Its great intensity makes possible visual examination, as well as photography with the highest-power objectives, while the exposures often last but a second or two and seldom, if ever, more than one minute, unless indeed colored screens be used. The Nernst lamp occupies an intermediate position between the Welsbach lamp and the arc-light both in regard to cost and excellence. The light proceeding from these various sources must, of course, be suitably collected and condensed by lenses, and in the case of the arc-lamp a cooling-cell (filled with water) must also be provided lest the heat of the focused rays cause injury to the objectives. Iris diaphragms and shutters are also frequently placed in the path of light so that the amount of it entering the objective may be controlled and greater sharpness secured.

The Camera.—For the taking of photo-micrographs a camera with the necessary light-tight connection, and advisably with Iris diaphragm and automatic shutter, must be provided. It

should be so constructed and disposed that connection with the microscope can be quickly made without disturbing any of the optical parts of the microscope or illuminating outfit.

In Fig. 9 the universal metalloscope is shown in connection with a Nernst lamp illuminating outfit and a vertical camera, while in Fig. 10 the metalloscope is illustrated with an electric-arc lamp and a camera in a horizontal position. There can be no doubt but the vertical is the correct position for the microscope, while the horizontal position offers serious advantages in case of the camera. Heretofore both microscope and camera had to be placed vertically or both horizontally, the microscopist having to put up with the inconveniences of a vertical camera or of a horizontal microscope. In the disposition shown in Fig. 10 this has been overcome by the use of a totally-reflecting prism of large dimensions fitted to the camera, and which can readily be brought over the eye-piece when it is desired to take a photograph, without disturbing any of the optical parts. The image is then sharply focused on the screen by turning a pulley fastened to the camera-standard near the screen and connected by a silk thread with the fine adjustment of the microscope.

The placing of the arc-lamp on the same side of the microscope as the camera is another important departure, making it possible for the operator while sitting at the screen to reach with his right hand the various adjustments of the lamp, thus securing maximum intensity and uniformity of illumination, two points so essential in taking photo-micrographs.

Apparatus for Metallography.

BY CARLE R. HAYWARD,* BOSTON, MASS.

(Wilkes-Barre Meeting, June, 1911.)

THE growing importance of metallography has caused a corresponding interest in the improvement of apparatus for preparing specimens of metals and alloys for microscopic examination.

The purpose of this paper is to describe an electric heating-furnace, a grinding- and polishing-machine, and a device for mounting specimens, which are used in the metallographical work at the Massachusetts Institute of Technology. These three pieces of apparatus were designed and made in the laboratory of the Institute, and each possesses some original features which may be of interest.

ELECTRIC HEATING-FURNACE.

The accurate control of heat necessary for metallographic work is best obtained by the electric-resistance furnace. Unfortunately, however, even the platinum-wound furnaces deteriorate with use and ultimately burn out, while the cheaper resistance-materials are often short-lived. Since the resistance-coil must be replaced from time to time, it is of advantage to be able to make this change with as little trouble as possible, and this is an important feature in the furnace shown in Fig. 1.

A is a cylindrical galvanized-iron can, with two handles, *B*. A porcelain tube, *C*, passes through the central hole in the cover of the can and rests upon the asbestos disk, *D*; the bottom of the tube is held in place by the asbestos ring, *E*. The space, *F*, around the tube is filled with powdered magnesia. *G* is an ordinary assay-crucible, with the cover inverted so as to present a smooth top for supporting a second porcelain tube, *H*. The latter is wound with "Excello" resistance-tape, 0.014

* Massachusetts Institute of Technology.

in. thick by 0.25 in. wide, the ends of which are shown at *I*. The spaces (0.1 in.) between the turns of tape are filled in with a paste made by moistening alumina with water-glass, which hardens and prevents contact between the adjacent coils. Both leads are insulated from the top of the can by asbestos cloth, and the one connecting with the bottom of the heating-coil is insulated by porcelain (not shown in sketch).

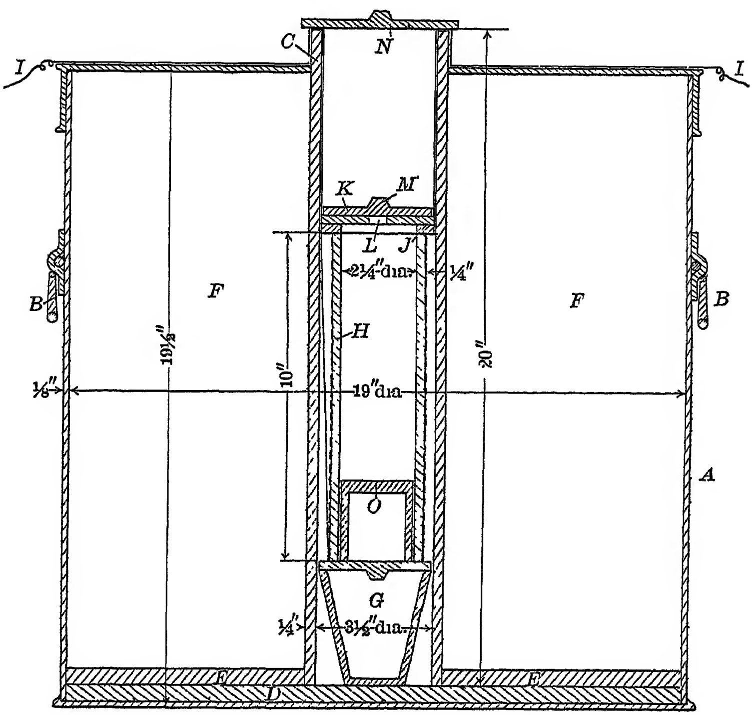


FIG. 1.—ELECTRIC HEATING-FURNACE.

An asbestos ring, *J*, hardened by a coating of water-glass, covers the space between the two porcelain tubes, and an asbestos disk, *K*, provided with a hole, *L*, for admitting the pyrometer-tube, serves as an inner cover to the heating-chamber. When the pyrometer is not in use, the clay covers, *M* and *N*, are placed in the positions indicated. The vessel in which the fusion is made is supported at the center of the heating-zone by the inverted cylindrical crucible, *O*.

A prepared inner tube is kept on hand for use in case of

accident to the one in the furnace; the exchange can be made in a few minutes.

Starting with the furnace cold, a temperature of 1,000° C. can be obtained in 2 hr. on the 110-volt circuit. The current is kept low (from 4 to 6 amperes) at first, to warm the tubes slowly, but it is gradually increased to 11 amperes, which is sufficient to maintain the temperature constant at 1,000° C. Since the furnace has been in use there has been no occasion to exceed this temperature, but there is no doubt but what 1,100° C. or more can be obtained. The coil now in use has been run intermittently for 240 hr. at 1,000° C., and about the same total time at 800° C.

Grinding- and Polishing-Machine.

Polishing specimens is always a vital subject for the metallographist, and any method which shortens the time or secures better results is of interest.

Several designs of mechanical polishers are on the market, the most common being the Sauveur machine, which has two disks revolving in a vertical plane, the faces of which allow the specimen to be prepared in four successive steps. The two main objections to this type of wheel, viz., the difficulty of applying the water and polishing-powder satisfactorily and the disadvantage of holding the specimen against the vertical wheel, can be overcome by using a disk revolving in a horizontal plane. Such a machine, used in the laboratory of the University of Wisconsin,[1] is said to give excellent results, as does the one recently installed in the laboratory of the Massachusetts Institute of Technology, shown in Figs. 2 and 3.

Fig. 2 shows a section of the machine with one of the polishing-disks in position. The disk, *A*, is made of cast aluminum, and to it is fastened a steel disk, *B*, by means of screws. These screws are counter-sunk, and the space above their heads is filled in with solder. The steel disk, and also the iron supporting-disk *C*, are tapped to fit the threaded head of shaft *D*. Each disk is provided with 10 iron points, *E*, for holding the cloth cover in place. To adapt the machine for grinding, disks *A* and *B* are removed and an emery wheel is fitted over the

[1] *Electrochemical and Metallurgical Industry*, vol. vii., No. 1, p. 15 (Jan., 1909).

top of shaft *D*, and fastened by means of an iron cap, and a bolt, which is screwed into the top of the shaft.

The power is transmitted by a belt to pulley *H*, keyed to horizontal steel shaft, *G*, and thence by bevel-gears, *F*, to vertical steel shaft, *D*. The vertical shaft is supported by a collar, cast in one with the iron frame, which is bolted to the iron base. The lower supports for the horizontal shaft form part of the base. The bearings are of babbitt metal. The sheet-iron shield, *I*, which is supported by the standards, *J* and

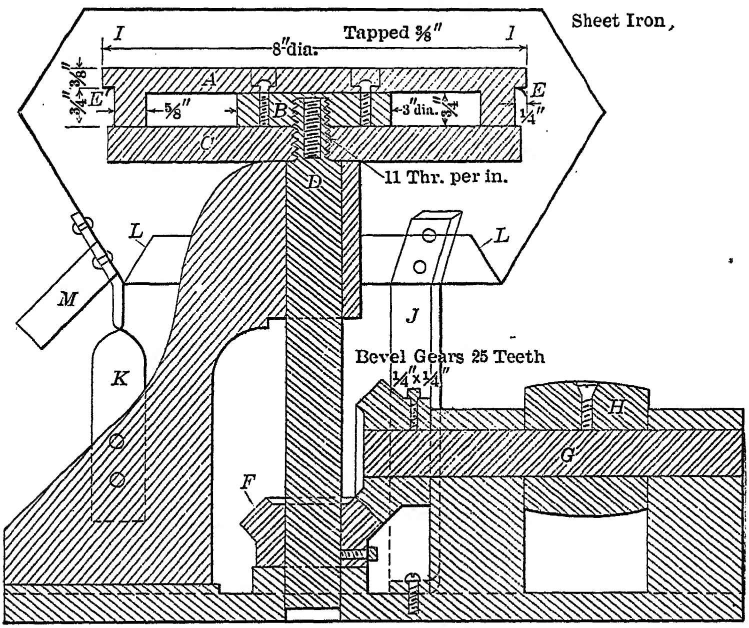


Fig. 2.—Section of Grinding- and Polishing-Machine.

K, has a trough, *L*, extending around the bottom, provided with a spout, *M*, to which a piece of hose may be attached for carrying off the water used in grinding and polishing.

Fig. 3 shows the machine in position for use. The polisher, *A*, and the motor, *B*, are bolted to the top of cabinet, *C*, which has a drawer for holding the grinding- and polishing-disks and a cupboard for extra cloths, bottles of polishing-materials, etc. On a shelf above the cabinet are placed four bottles, *D*, one of which contains clear water, and the others, water holding in sus-

pension flour-emery, tripoli-powder, and rouge, respectively. Through one of the three holes in the rubber stopper is passed a glass tube, *E*, to admit a jet of air for keeping the polishing-powder in suspension. Through another hole passes the glass tube, *F*, which, with the rubber tube, *G*, acts as a siphon for carrying the solution to the polishing-disk. The rubber tube is supported by slots in the wooden arms, *H* and *I*, and termi-

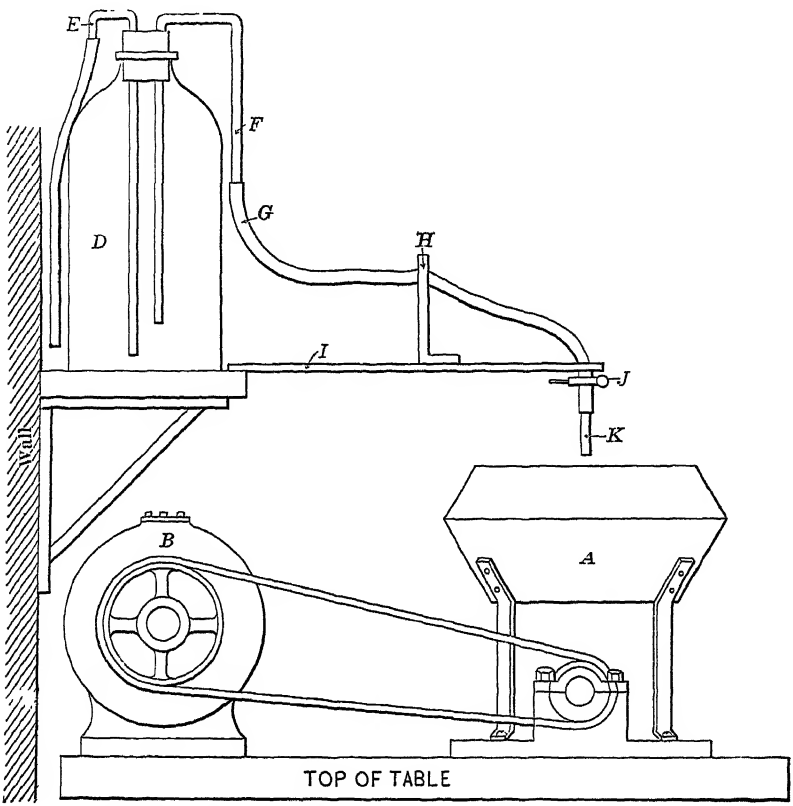


Fig. 3.—Grinding- and Polishing-Machine in Position for Use.

nates in a glass tip, *K*, directly above the center of the disk. The flow of the solution is regulated by the pinch-cock, *J*.

The mode of operation is as follows:

The emery-wheel is put in position, the tube from the water-bottle placed on the arms, *H* and *I*, Fig. 3, and the motor started. The disk rotates at 1,800 rev. per min., and quickly grinds a level surface on the specimen. The motor is

then stopped, the emery-wheel removed, and the aluminum disk, covered with canvas, put in position, Fig. 2. The tube from the bottle containing flour-emery suspended in water is now placed on the arms, *H* and *I*, and the motor started. This process is repeated with two more aluminum disks covered with broadcloth and kept moistened with solution from the tripoli-powder and rouge bottles respectively. The specimen may be finally cleaned on the rouge-wheel by moistening with clear water instead of rouge emulsion.

The method used for attaching the aluminum disks to the shaft leaves the entire surface available for polishing, which is not possible when a cap is used such as is required by the emery-wheel.

New cloths may be easily placed on the aluminum disks by simply stretching them over the surface and hooking them on the pins, *E*, Fig. 2.

The disks are easily removed, and when not in use are kept in the cabinet away from dust.

The machine, as described above, was designed to be belted to shafting, and where several machines are required, the expense of separate motors may be obviated by doing this; but where only one machine is necessary, it would undoubtedly be more satisfactory to dispense with the gears and horizontal shaft, and couple the upright shaft directly to a motor with a vertical armature.

Other improvements and modifications would undoubtedly suggest themselves after longer use.

Mounting-Device.

In order to obtain good results in making photo-micrographs, the surface of the specimen must be perpendicular to the axis of the microscope. Various devices for obtaining this end have been proposed from time to time, each of which has some good features. The specimen-mounter used in the laboratory of the Massachusetts Institute of Technology, Fig. 4, is a modification of that recommended by G. H. Gulliver.[2] A piece of 2-in. iron pipe, *A*, is threaded upon the iron standard, *B*, the upper surface of which is made parallel with the plane of the

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[2] *Metallic Alloys* (Lippincott, Philadelphia, 1908).

top of the pipe. When the apparatus is to be used, the top of the standard is covered with a clean piece of paper and the specimen placed face down upon it; the pipe is then turned up or down until the distance between the highest point on the base of the specimen and the plane of the top of the pipe is about $\frac{1}{8}$ to $\frac{1}{16}$ in. Finally, a piece of glass, *C*, to which is attached some plastic wax, *D*, is placed above the specimen, *E*, and pressed down upon the top of the pipe, thus imbedding

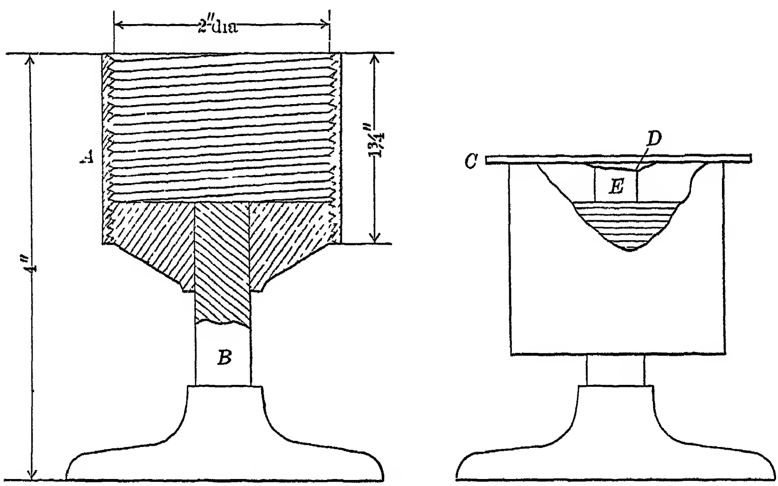

FIG. 4.—DEVICE FOR MOUNTING SPECIMENS.

the base of the specimen in the wax. It is evident that the surface of the specimen thus mounted is parallel to the glass base, and therefore will be parallel to the stage of the microscope. Such a mounting-device is easily constructed, and gives excellent satisfaction as an accessory to a vertical microscope, as it can be quickly adjusted to take specimens having different sizes.

Biographical Notice of Samuel Franklin Emmons.

BY GEORGE F BECKER, WASHINGTON, D C.

(San Francisco Meeting, October, 1911.)

A MERE record of Emmons's professional career would very inadequately represent the man. That he was eminent we know, and our successors will realize in due time; but they must depend upon us for knowledge of a singularly wholesome, modest, unselfish personality, and of a character that did honor to a profession in which trustworthiness is indispensable. Those members of the Institute who met Emmons are his friends, and I never knew one of these who was not the better for that friendship.

Emmons was born in Boston, Mar. 29, 1841, the son of Nathaniel H. and Elizabeth (Wales) Emmons, and was named Samuel Franklin after an ancestor who was of the same family as Benjamin Franklin. He took the degree of Bachelor of Arts at Harvard in 1861, and soon afterwards went abroad to complete his education. From 1862 to 1864 he attended the courses at the École Impériale des Mines at Paris, Élie de Beaumont and Daubrée being among his professors. The year 1864–1865 he spent at Freiberg under Cotta and other famous teachers; after which he spent another year in traveling through Europe. Like many other renowned geologists, he approached his ultimate profession from its economic side, and was thus from the first imbued with a sense of high responsibility in the promulgation of scientific opinions or conclusions. With hypotheses which were interesting merely because they were ingenious or even plausible, he would have nothing to do.

In 1867 he joined the Geological Exploration of the Fortieth Parallel at its organization under Clarence King, serving at first as a volunteer, but soon receiving a regular appointment. This expedition was the first one of purely geological character sent out by the United States government. As Emmons has shown in his admirable presidential address on "The Geology of Government Explorations," its work was founded on a com-

plete and comprehensive plan, adopted before taking the field, and systematically followed in all essential features during the ten years of its existence. This plan aimed at the highest efficiency compatible with prompt completion. It was important from every point of view that the broad outlines of the geology and mineral resources of the belt of country to be opened up by the completion of the transcontinental railway should be made known as soon as practicable. To execute a final, detailed survey under such conditions was impossible; and for this reason the work was called an exploration, but as a first approximation to the truth the intention was to make it irreproachable in methods and in symmetry. The best experts to be had were secured; contour-mapping as a basis for geological work was introduced for the first time in this country;[1] and, when lithological collections had accumulated, a well-known European petrographer was engaged to discuss them by the new microscopic methods, then wholly unfamiliar to American geologists. Emmons's associates as assistant geologists were our late eminent colleague James D. Hague, and his brother, Mr. Arnold Hague. The expedition started in 1867 from Sacramento; and it will help our younger brethren to grasp the changes which have taken place in the civilization of the West to be reminded that an escort of 30 regular soldiers was needed in that year to protect the civilians from hostile Indians.

To realize how hard the men worked, it is only needful to glance at the Fortieth Parallel memoirs and maps, but shooting was an available recreation, and afforded a legitimate means of varying a monotonous diet. There was one particularly good bear-story which Dr. Raymond has recorded in his notice of King in Emmons's own words.[2] Of this Dr. Raymond writes me:

> "King, who always reaped the glory which his splendid audacity deserved, killed the bear; but the story shows that Emmons was posted at the other end of the passage where the wounded bear would have come out, if King's shot in the dark had not been fatal!"

One of the rules of the Fortieth Parallel Survey was, that its

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[1] Capt. John Mullan's Report on the Construction of a Military Road, 1863, contained contour sketch-maps surveyed and drawn by Theodore Kolecki.

[2] *Trans.*, xxxiii., 633 (1903).

members should be as civilized as practicable, especially at meals. The men believed in a good and varied diet, well-cooked and served; and, when the accounting-officers of the War Department demurred at passing a bill for currant-jelly, they were met with a threat to charge up at the rate of beef the venison furnished by members of the mess. By such means the geologists preserved both their digestion and their adaptability to social life at centers of civilization, in which every one of them took a prominent part in later years.

Two episodes in the history of the Exploration of the Fortieth Parallel deserve mention. In 1870, the appropriation-bills passed too late for a regular season of field-work, and King decided on an examination of the extinct volcanoes of the Cascade range. He and Emmons ascended Mt. Shasta, and there found the first glaciers recognized within the limits of the United States. Later, in the same autumn, Emmons made the ascent of Mt. Rainier, where he found much more extensive glaciers, which he has very graphically described. Emmons made no claim to the first ascent of this great peak, recognizing that it had been scaled two months earlier by Gen. Hazard Stevens; but our colleague, who was accompanied by Mr. A. D. Wilson, was the first to bring from this dormant volcano valuable information on its topography, geology, and glaciology. During the same season Mr. Arnold Hague ascended Mt. Hood, where he too discovered typical glaciers.

In 1872 Emmons took part in the exposure of the famous diamond swindle. Though strong efforts were made to keep secret the locality of the alleged diamond "discovery," King made out that it must be in a region which Emmons had surveyed. They set out together to investigate, and Emmons was able to lead the little party to the scene of the crime in Vermillion Creek Basin, Wyoming. This had been selected by the swindlers because it was in a nearly waterless region, from which almost any expert would retreat at the first possible moment. A great financial disaster was averted by the detection of this fraud, and it is doubtful whether King could have achieved the disclosure without Emmons's knowledge of the country.

In King's Exploration, Mr. Arnold Hague and Emmons had charge of the descriptive geology. In 1870 Emmons had con-

tributed a chapter on the Toyabe range and some minor notes to Vol. III. (Mining Geology). With these exceptions, and that of his description of Mt. Rainier, all his work on that survey is contained in Vol. II. (Descriptive Geology), printed in 1877, and containing nearly 900 pages. In the letter of transmittal by the authors the limitations of the work are thus emphasized :

"It will be readily understood by the reader, from the very title of the work, that this does not claim to be a systematic survey like those of Europe, based on accurate maps, but is rather a geological reconnaissance in an unknown and often unexplored region, where geology and topography had to go hand in hand, and that therefore, while details were often, from the necessities of the case, somewhat neglected, it was the general bearing of the leading geological facts that was most constantly in our minds."

Now-a-days, I suppose, no one would think of reading this volume through, though it remains an important book of reference. In 1877, however, it was full of news, and Gerhard vom Rath, to whom geology (directly and indirectly) owes so much, told me in 1883 that it was the interest the Descriptive Geology aroused in him which led him to visit the United States. It was, I remember, the first work I ever reviewed; and I greatly enjoyed the task.

In accordance with the plan of the Exploration of the Fortieth Parallel all the men had constantly to guard against two temptations, one being to follow out their problems by detailed studies at an undue expenditure of time, and the other to gain time by slighting important matters in which they might happen to feel relatively slight personal interest. There can be no doubt that they displayed great self-control; and in my opinion the result was an unrivaled model of a preliminary survey in an unknown region.

It should not be forgotten that the topographic and the geologic reconnaissances were executed at substantially the same time, so that the geologists rarely had maps in the field on which to record their work. This involved keeping in mind and in note-books a vast number of detailed observations systematically co-ordinated and of a prescribed standard in respect to generality. Ten years of this sort of thing gave Emmons an unusual command of details, and power to marshal them mentally without extraneous aid. In short, it was the

training in descriptive geology, as he practiced it, which enabled him subsequently to deal with the complexities of Leadville.

With the completion of the Descriptive Geology in 1877, the connection of its authors with the Fortieth Parallel ceased. For the next two years, Emmons devoted himself to a cattle-ranch near Cheyenne. The country was still unfenced, and great profits were possible in this business, while the active, out-door life suited Emmons's temperament and habits. Even after his return to scientific life, he retained his interest in this ranch for some years, and kept there a pack of Scotch deer-hounds with which he hunted.

In March, 1879, the government organizations which had been carrying on geological reconnaissances were merged in the present United States Geological Survey, and King was appointed Director, taking his oath of office on May 24. As a matter of course, a position was offered to Emmons, and he qualified on August 24.

In the autumn of that year King summoned Emmons and me to Washington, in order to prepare schedules for the examination of the precious metal industries under the Tenth Census, a task undertaken by the Survey as a matter of courtesy to the Census Bureau and as germane to its own office. As soon as Emmons arrived, I called upon him; and when, an hour later, King entered the room to introduce us, we were already friends. Such we always remained without a single misunderstanding.

It was for each of us a busy and interesting period, and in later years a favorite subject for reminiscence. Emmons, though of course strong on general geology and lithology, was rusty in technical mining and metallurgy, which I had been teaching; and while I had a considerable familiarity with ore-deposits, my knowledge of general geology and lithology was elementary. Indeed, on joining the Survey, I had stipulated with the Director that he should call upon me only for mining and metallurgical reports. Thus the preparation of schedules led to many instructive discussions, carried on with the utmost freedom and good-will. We worked hard and long. We were in almost daily consultation with King, who was well informed on the whole subject, but I do not remember

that he ever made any material change in our plans; and we also had prolonged sessions with Gen. Francis A. Walker, Superintendent of the Census, who was thoroughly agreeable and agreeably thorough.

Life was not all work, however. John Hay, then Assistant Secretary of State, and King had at Wormley's a private dining-room, which they invited Emmons and me to share with them. I doubt whether there ever was table-talk more brilliant than that to which we listened in that room. Neither Emmons nor I said much, but we egged on the other two, and laid little plots to get them started on matters we desired to hear discussed. King and Hay were intimate friends, and particularly well-fitted by differences in temperament and experience to complement one another in conversation. Though Hay rarely indulged in humor and was not a man of buoyant spirits, he was never commonplace or ponderous. He was gifted with true wit, whose gleams showed even familiar relations in new aspects and revealed relationships among less familiar things. He offered, but never obtruded, suggestive reflections gracefully epitomized, and in this intimate companionship disclosed the grasp of affairs and breadth of view which were to make him a great Secretary of State. As for King, hear Hay!

> "He was inimitable in many ways: in his inexhaustible fund of wise and witty speech; in his learning, about which his marvellous humor played like summer lightning over far horizons; in his quick and intelligent sympathy, which saw the good and amusing in the most unpromising subjects; in the ease and airy lightness with which he scattered his jewelled phrases; but above all in his astonishing power of diffusing happiness wherever he went."[3]

Had those wonderful dinners not been so entertaining they might have been considered as equivalent to a post-graduate course in liberal education. They exerted a lasting influence on Emmons and me, expanding our views and adding symmetry to our standards of thought and achievement.

It was while we were engaged on the Census schedules that King completed his plans for the investigation of ore-deposits, and placed the work in our charge by the orders quoted in Emmons's introductory chapter to this volume. I was reluctant to accept the responsibility, and I should have persisted in

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[3] *Clarence King Memoirs*, p. 131 (1904).

refusing it, had not Emmons urged me to make the attempt, assuring me in the kindest manner of his co-operation and assistance, so far as circumstances might permit. He began to cram me immediately; and, during the field-work in Leadville and on the Comstock, we were in constant correspondence on every phase of both problems.

Early in 1880, each of us had to select and instruct a staff of young mining engineers who were to collect the statistics and technological data under the Census, while at the same time we organized and commenced our geological field-work; in fact Emmons began on the geology of Leadville just before the New Year.

What little is to be said of the Census work may be said here, although it was not completed until Emmons's abstract of his Leadville report had been printed. The purpose of the Statistics and Technology of the Precious Metals was to furnish mining-men with accurate data of production and a record of technical practice in the year 1880, together with such an outline of the geology of the mining-districts as could be prepared from material already published, supplemented by the information derived from the reports and collections sent in by the experts in the field. It was a harassing piece of work; and it is needless to say that some districts were more competently reported than others; but under the circumstances, and on the whole, the authors were fairly satisfied with the result. Its value would have been enhanced by prompt publication. By working at night and on holidays the manuscript and maps were completed and transmitted on Feb. 8, 1883; but more than a year elapsed before the first galley-proofs reached us; and in the meantime the maps had disappeared from the Census Office. I remember exactly how we felt!

After King retired from government work, I was placed in charge of the Statistics and Technology of the Precious Metals, so that for a time I had the honor of counting Emmons nominally as my assistant; but of course we worked together as before, and no question of subordination was allowed to arise. When it came to deciding the order of our names on the title-page, however, he said I was in charge and should come first, while I maintained that he, as the senior and more experienced, should take precedence. As neither would be convinced, I

proposed deciding the matter by the turn of a coin. Thus we settled it, standing by the statue of Jackson, in the city of Washington. He won the toss and I my way.

In spite of the labor involved in gathering statistics under the Census, Emmons pushed the examination of the geology of Leadville so energetically that he was able to close his office at the camp on Apr. 1, 1881, and to transmit his Abstract of a Report on the Geology and Mining Industry of Leadville on October 20 of that year. This abstract, which appeared in the *Second Annual Report of the Director of the United States Geological Survey*, is a memoir of 87 pages and contains the principal results of the investigation. The publication of the *Monograph* was delayed by various causes till 1886; but his main conclusions were not changed in the intervening time.[4] In the field-work he was assisted by Ernest Jacob, Whitman Cross, and W. H. Leffingwell as geologists, and by W. F. Hillebrand and Antony Guyard as chemists.

Emmons's views of the Leadville ore-deposits, up to the time of the publication of his *Monograph*, may be condensed into the following statement: Prior to oxidation, the ores consisted of sulphides of lead and silver, zinc and iron, which were deposited by substitution for country-rock, this being as a rule limestone or dolomite, but in some instances siliceous in character. The ore reached the deposits as hot aqueous solutions at high pressures, and came from above. The temperature was due to the depth (about 10,000 ft.), and the magmatic heat of the intrusive porphyries. The water was of meteoric origin and derived its metallic content, perhaps wholly but demonstrably in part, from masses of porphyry which were not necessarily in juxtaposition with the ore. The principal deposition took place at the upper surface of the blue Carboniferous limestone.[5]

Twenty-one years later he returned to the subject with Mr. J. D. Irving in a paper on the Downtown District; [6] and the only

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[4] In the *Abstract* Emmons regarded the ore as derived from the porphyries, while in the *Monograph* he considered them as "mainly" derived from this source.

[5] See Abstract, *Second Annual Report, U. S. Geological Survey*, p 234 (1882). Geology and Mining Industry of Leadville, p. 584 (1886). Also, *Trans.*, xv., 138 (1886–87).

[6] *Bulletin No.* 320, *U. S. Geological Survey* (1907).

important change he was obliged to make was an addition rather than a correction. Developments in the intervening decades had shown that many, instead of few, ore-bodies existed within the mass of the Carboniferous limestone (not merely near its upper surface), and also in the Silurian limestone. Meanwhile, however, the subject of juvenile or magmatic waters, first investigated by Charles Sainte-Claire Deville and other French *savants*, had been actively studied and discussed, so that in 1907 questions arose as to the possible participation of such waters in the genesis of the Leadville deposits. How far the original sulphides at Leadville were deposited from juvenile waters, and whether instances of deposition as a feature of contact-metamorphism were to be found there, were still unknown.

This paper by Emmons and Irving was, in fact, a partial abstract in advance of a monograph by the same authors, in which the entire Leadville work was to be revised. Fortunately the volume was so nearly completed before the senior author's death that Mr. Irving is in a position to finish it within a few months. How far it will answer the questions which were still open in 1907, I do not know.

.Leadville presents the most intricate problem of mining-geology ever attempted; for the structure is as complex as the chemical history of the deposits. Emmons brought to the study of this district a mind trained to carry a vast number of observations in due relation to one another; and this enabled him to execute a truly monumental work. His *Monograph* has been of enormous importance to miners, for experience has shown that its predictions were substantially correct; it has been of material advantage to the Geological Survey as an evidence of what geology can do for industry; and it has set an example to younger geologists of the mode of treatment proper to such a problem. The revision of this great work after the lapse of 30 years worthily closed his career.

Having concluded that the Leadville ores were deposited by substitution, mainly for limestone, Emmons was led to study instances of the replacement by ore of other rocks. Indeed, even in his Leadville Abstract of 1882, he had recognized limited occurrences of ores substituted for siliceous rock. Cases of this kind had been described in Europe by Groddeck

and others; but in this country only the native copper of Lake Superior had been recognized as pseudomorphic by Mr. Pumpelly. Emmons soon found abundant evidence capable of interpretation as indicating replacement or metasomatism in a wide sense; that is to say, he found much ore in situations from which even siliceous rocks or minerals had been removed. To cover them all, he defined metasomatism as an interchange of substances, but not necessarily molecule by molecule.

This breadth (perhaps I ought to say looseness) of definition was unavoidable unless he had been willing to postpone for years the announcement of his results; for convincing detailed proof of the various processes active in the alteration and impregnation of wall-rocks requires prolonged and difficult chemical and microscopical investigation. Among engineers the idea, new to many of them, immediately became popular, too popular in fact; and at one period there was danger that all deposits would be set down without due proof as cases of replacement. Some, however, were left to protest; and, after a few years, the matter was reduced to proper proportions by Mr. Lindgren,[7] who, adopting as his criterion the principle that the theory of substitution of ore for rock is to be accepted only when there is definite evidence of pseudomorphic, molecular replacement, worked out his results with great labor and discrimination. There can be little doubt that as geological chemistry is elaborated the importance of deposition by substitution will be still further recognized, and that studies devoted to this subject will shed unexpected light on geochemical processes.[8]

Secondary enrichment of sulphide ores attracted atten-

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[7] *Trans.*, xxx., 596 (1900).

[8] Among the very first observations which I made on the Comstock lode was, that much of the pyrite in the wall-rock was pseudomorphic after ferromagnesian bisilicates. (*Geology of the Comstock Lode*, p. 210, 1882.) Emmons's studies on replacement led me to examine the quicksilver-mines very closely for pseudormophic deposition of cinnabar. In spite of profound alteration of wall-rock, attended by other replacements, I found no instance of deposition of cinnabar by substitution for carbonates or silicates. These facts led me to suggest the dialytic or osmotic separation of ore-bearing solutions, a hypothesis which is thus indirectly due to Emmons. *Geology of the Quicksilver Deposits of the Pacific Slope*, p. 396 (1888), and *Mineral Resources of the U. S. for* 1892, U. S. Geological Survey, p. 156 (1893).

tion in Europe earlier than in this country. The relative affinity of the metals for sulphur was investigated as long ago as 1837, by E. F. Anthon,[9] but the first application to ore-deposits with which I have met is contained in Mr. Joaquin Gonzalo's admirable monograph on Huelva, issued in 1888. The secondary deposits of chalcopyrite (occasionally accompanied by other copper-compounds), and galena, as they are found at Rio Tinto, are described by the Spanish geologist as occurring along lithoclastic fractures in the mass of the pyrite. They are attributed to a process of segregation within the mass and to the reduction of sulphates percolating downward from the zone of oxidation.[10] Mr. J. H. L. Vogt, after personal examination, entirely assented to Mr. Gonzalo's views, and pointed out subsequently that secondary enrichment is the true meaning of that familiar old proverb: *Es thut kein Gang so gut, Er hat einen eisernen Hut.*[11]

Emmons's own studies on secondary enrichment were begun at Butte in 1896; and he freely discussed his results in private, though they were first published in our *Transactions* in 1900. In this paper, he quotes from that of Vogt, issued the year before, but also sets in order a long series of observations of his own, which form an extremely important contribution to the subject. This is cognate to his other studies on replacement; for his idea of secondary enrichment might be paraphrased as the replacement of pyrite by the sulphides of other metals, especially copper.

The idea of secondary enrichment was in the air at the close of the last century, and had been very distinctly suggested in this country (for instance by Dr. James Douglas), though without sufficient substantiation. Almost simultaneously with Emmons's memoir appeared important papers by Messrs. Weed, Van Hise, and Lindgren.

It is not needful here to pass in review all of Emmons's work. A full list of his papers will be found at the end of this notice. All of them are as conscientiously elaborated as those

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[9] *Journal für praktische Chemie*, vol. x., p. 333 (1837). See also E. Schürmann, in *Liebig's Annalen*, vol. ccxlix., p. 326 (1888).

[10] Mem. de la Comm. del Mapa Geológica de España. Descripción física, geológica y minera de la provincia de Huelva, por Joaquín Gonzalo y Tarín, pp. 217 to 220 *et passim* (1888).

[11] *Zeitschrift für praktische Geologie*, pp 241 to 254 (July, 1899).

which I have selected for mention on account of their peculiar importance. On the other hand, a few remarks seem appropriate on the tendency and the development of the science which he so admirably represented.

When Clarence King planned the researches of the U. S. Geological Survey into the origin and nature of ore-deposits, and placed Emmons and me in charge of them, no one of us was in a position to appreciate the multifariousness and intricacy of the facts which these investigations would disclose; but before King's untimely death, the vastness of the task was manifest, as well as the necessity for improved methods of investigation and for experimental researches of the most fundamental character.

More than half of the great amount of information now available to mining-geologists is due to the use of the microscope, armed with which, the eyes of the generation now passing away have been a hundred times as sharp as those of their predecessors. But the microscope is not merely a powerful magnifying-glass; it is an instrument of moderate precision, whose use has familiarized us with quantitative measurement and stimulated us to demand exact methods of geological investigation.

It is not enough to know the facts, for these alone lead only to delusive "rules of thumb." We can and must attain a comprehension of the mechanical, chemical, and thermal processes which underlie the formation and distribution of ores, as revealed not only by the microscope, but also by every other available method of research. Many of the problems presented are of extraordinary difficulty, far exceeding in this respect most of those undertaken by professional physicists and chemists; but they are not insoluble; and the limits of our knowledge are extended year by year.

None of us have been more impressed with the necessity for such researches than was King, or even Emmons, who regretted all his life that he had not a better command of the exacter sciences. Let me pass the word on from them that the future of the science of ore-deposits depends on investigations of the utmost precision into the fundamental principles of geophysics. Physics and physical chemistry will be as indispensable to the mining-geologist of the future as mineralogy to the petrographer or zoölogy to the palæontologist. It is a duty which

the Institute owes to its founders, its members and the world, to promote and foster research of this description; to advance as rapidly as possible the day when mining-geologists, no longer groping, will comprehend why ore-deposits are what we find them to be.

And now as to the man himself. There is not a geological society or even a mining-camp from Arctic Finland to the Transvaal, or from Alaska to Australia, where Emmons's name is not honored and his authority recognized; nor is there a society of which he was nominally an active member in which he was not really active and efficient. Thoroughness and good judgment characterized all he did. He had a very high sense of responsibility and rarely made his hypotheses public; yet his originality has enriched the science to which his life was devoted. In private life, he was modest to the point of diffidence, and many of his old acquaintances scarcely knew of his distinction; but none could long enjoy his acquaintance without becoming conscious of the kindness of his heart and the elevation of his character. He would not have known how to undertake an unworthy action, or how to do a selfish thing. His published investigations will live on as sources of knowledge and models of method; and in a smaller circle his personal example will continue potent for good.

Emmons died painlessly and unexpectedly in his sleep on Mar. 28, 1911, the eve of his seventieth birthday. Thus fitly ended a career of useful labor faithfully performed.

Among the societies to which Emmons belonged, none appealed to him more than the American Institute of Mining Engineers. He joined us in 1877, was three times Vice-President, contributed many papers to the *Transactions*, and was always ready to assist in organizing our meetings. He was also a member of the National Academy of Sciences, the American Philosophical Society, the American Academy of Arts and Sciences, the Washington Academy of Sciences, the Geological Society of London, the Geological Society of America, the International Congress of Geologists, and the Colorado Scientific Society. He was elected an honorary

member of the Société Helvétique des Sciences Naturelles, and received the degree of Doctor of Sciences from Harvard and Columbia.

List of Scientific Publications of Samuel F. Emmons.

1870. Geology of Toyabe Range.
U. S. Geol. Exploration of 40th Parallel. Vol. III. Mining Industry. Chap. VI, sect. II, pp. 330–348, with colored geological map.

Geology of Philadelphia or Silver Bend region.
Ibid., chap. VI, sect. II, pp. 393–396.

Geology of Egan Cañon District.
Ibid , chap. VI, sect. VI, pp. 445–449.

1871. Glaciers of Mt. Rainier.
Amer. Jour. Sci., 3d ser. Vol. I, pp. 161–165.

1877. The Volcanoes of the U. S. Pacific Coast.
Address delivered at Chickering Hall, N. Y., Feb. 6, 1877. Jour. Amer. Geogr. Soc. Vol. IX, 1876–'7, pp. 44–65.

1877. Descriptive Geology of the 40th Parallel.
U. S. Geol. Exploration of 40th Parallel. Vol. III (with Arnold Hague). 4to, pp. 850, with 26 plates and atlas of 11 maps and 2 section sheets, colored geologically.

1882. Abstract of Report on Geology and Mining Industry of Leadville, Colo.
U. S. Geol. Survey, Second Ann. Report. Pp. 203–290, with geological colored map and sections.

1882. The Mining Work of the U. S. Geological Survey.
Trans. Am. Inst. Mg. Eng'rs. Vol. X, pp. 412–425.

1883. Geological Sketch of Buffalo Peaks.
U. S. Geol. Survey, Bulletin No. 1, pp. 11–17.

1883–4. Opportunities for Scientific Research in Colorado. Presidental addresses.
Proc. Colo. Sci. Soc. Vol. I, pp. 1–12 and 57–61.

1883. Ore Deposition by Replacement.
Proc. Phil. Soc. Wash'n. Vol. VI, p. 32.

1884. What Is a Glacier?
Proc. Phil. Soc. Wash'n. Vol. VII, p. 37.

1885. Statistics and Technology of the Precious Metals.
Tenth Census Reports. Vol. XIII (with G. F. Becker); Gov't 4to, 541 pages. (Submitted in 1883.)

1886. Geology and Mining Industry of Leadville, Colo.
U. S. Geol. Survey, Monograph XII, 779 pages and 45 plates, with atlas of 35 sheets of maps and sections colored. (Submitted in 1885.)

1886. Genesis of Certain Ore-Deposits.
Trans. Am. Inst. Mg. Eng'rs. Vol. XV, pp. 125–147.

1886. Notes on Some Colorado Ore Deposits.
Proc Colo. Sci. Soc. Vol II, pp. 85–105.

1887. On the Origin of Fissure Veins.
Ibid., Vol. II, pp. 187–202.

1887. On Glaciers in the Rocky Mountains.
Ibid , Vol. II, pp. 211–227.

1887. Preliminary Notes on Aspen, Colo.
Ibid., Vol. II, pp. 251–277.

1887. Submerged Trees of the Columbia River.
Science. Vol. XX, pp. 156–157.

1887. Notes on the Geology of Butte, Mont.
Trans. Am. Inst. Mg. Eng'rs. Vol. XVI, pp. 49–62.

1888. Structural Relations of Ore-Deposits.
Ibid., Vol. XVI, pp. 804–839.

Same translated into French by R. A. Bergier.
Révue Universelle des Mines. Tome X, 3me ser., 34me ann., p. 130. Liège et Paris, 1890.

1888. On Geological Nomenclature.
Rep. of Am. Com'te Intern. Congress of Geologists, pp. 58–61.

1889. On Orographic Movements in the Rocky Mountains.
Bull. Geol. Soc. Am. Vol. I, pp. 245–286.

1890. Age of Beds in the Boise River Basin, Idaho.
Proc. Bost. Soc. Nat. Hist. Vol. XXIV, pp. 429–434.

1890. Notes on Gold-Deposits of Montgomery County, Md.
Trans Am. Inst. Mg. Eng'rs. Vol. XVIII, pp. 391–411.

1892. Fluorspar-Deposits of Southern Illinois.
Ibid., Vol. XXI, pp. 31–53.

1892. Faulting in Veins.
Eng'r'g and Min'g Journal. Vol. LIII, pp. 548–549.

1893. Compte Rendu de la 5me Session du Congrès Géologique Internationale (Editor).
Gov't Printing Office. 529 pages, 21 plates, 39 figures.

1893. Geological Distribution of the Useful Metals in the United States.
Trans. Am. Inst. Mg. Eng'rs. Vol. XXII, pp. 53–95.

1893. Genesis of Ore-Deposits (discussion).
Trans. Am. Inst. Mg. Eng'rs. Vol. XXIII, pp. 597–602.

1893. Progress of the Precious Metal Industry in the U. S.
U. S. Geol. Survey. Mineral Resources for 1902, pp. 46–94; also in Report of the Director of the Mint for 1893, pp. 117–141.

1894. Geological Guide Book to the Rocky Mountains.
John Wiley & Sons, New York.

1894. Geology of Lower California (with G. P. Merrill).
Bull. Geol. Soc. Am. Vol. V, pp. 489–514.

1894. Geology and Mineral Resources of the Elk Mountains, Colo.
U. S. Geol. Survey. Folio 9. Explanatory text.

1895. Geology of the Mercur Mining District, Utah.
U. S. Geol. Survey. 16th Ann. Report, pp. 349–369.

1896. Geological Literature of the South African Republic.
Journal of Geology. Vol. 4, pp. 1–22.

1896. Some Mines of Rosita and Silver Cliff, Colo.
Trans. Am. Inst. Mg. Eng'rs. Vol. XXVI, pp 773–823.

1896. The Mines of Custer County, Colo.
U. S. Geol. Survey. 17th Ann. Report, part II, pp. 411–472.

1896. Geology of the Denver Basin in Colorado (with W. Cross and G. E. Eldridge).
U. S. Geol. Survey, Monograph XXVII, 4to, 526 pages, with 31 plates, 102 figures.

1897. The Geology of Government Explorations (Presidential address before the Geological Society of Washington, December, 1896).
Science, n. s. Vol. V, pp. 1–15 and 42–51.

1897. Economic Geology of the Butte District, Mont.
U. S. Geol. Survey. Folio No. 38. Explanatory text.

1897. Physiography of the West Coast of Peru, S. A.
Science, n. s. Vol. V, p. 889.

1897. The Origin of Green River.
Science, n. s. Vol. VI, pp. 19–21.

1898. Geology of the Ten-mile District, Colorado.
U. S. Geol. Survey. Geol. Atlas of the U. S., Folio No. 48. Explanatory text.

1898. Map of Alaska: Its Geography and Geology.
U. S. Geol. Survey. 44 pages and geol. maps. Special report to the Fifty-fifth Congress, 2d session.

1898. Geology of the Aspen Mining District, Colorado.
U. S. Geol. Survey, Monograph XXXI, pp. XVII–XXXII. Introduction.

1898. Dr. Don's Paper on the Genesis of Certain Auriferous Lodes (discussion).
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1898. A Century of Geography in the United States.
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1898. Geological Excursion Through Southern Russia.
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1899. Plutonic Plugs and Subtuberant Mountains.
Geol. Soc. Wash'n, and abstract in Science, n. s. Vol. X, pp. 24–25.

1900. Secondary Enrichment of Ore-Deposits.
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Trans. Am. Inst. Mg. Eng'rs. Vol. XXXI, pp 658–683.

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1901. Clarence King—A Memorial.
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1902. Biography of Clarence King.
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1902. The U. S. Geol. Survey in its Relation to the Practical Miner.
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1902. Sulphidische Lagenstätten vom Cap Garonne.
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1902. On the Secondary Enrichment of Ore-Deposits (discussion).
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1902. On the Hydrostatic Level Attained by the Ore-Depositing Solutions in Certain Mining Districts of the Great Salt Lake Basin (discussion).
Trans. Am. Inst. Mg Eng'rs. Vol. XXXIII, p. 1062.

1903. Reminiscences of Clarence King.
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1903. Drainage of the Valley of Mexico.
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1903. Little Cottonwood Granite Body of the Wasatch Mountains.
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1903. Contributions to Economic Geology, 1902 (introduction).
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1904. Contribution to Economic Geology, 1903. Metalliferous Ores.
U. S. Geol. Survey, Bull. No. 225, pp. 18–24.

1904. Economic Resources of the Northern Black Hills, by J. D. Irving, with contributions by S. F. Emmons and T. A. Jaggar, Jr.
U. S. Geological Survey, Prof. Paper No. 26. 222 pp.

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1905. Investigation of Metalliferous Ores.
U. S. Geol. Survey, Bull. No. 260, pp. 19–27.

1905. Copper in the Red Beds of the Colorado Plateau Region.
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1905. The Cactus Copper Mine, Utah.
U. S. Geol. Survey, Bull. No. 260, pp. 242–248.

1905. Contributions to Economic Geology, 1904.
In U. S. Geol. Survey, Bull. No. 260.

1905. Economic Geology of the Bingham Mining District, Utah, by J. M. Boutwell; with a section on areal geology by Arthur Keith, and an introduction on general geology by S. F. Emmons.
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1906. A Map and a Cross Section of the Downtown District of Leadville, Colo.
Abstract: Science, n. s. Vol. XXIII, pp. 816–817.

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1906. Los Pilares Mine, Nacozari, Mexico.
Econ. Geol. Vol. I., No. 7, pp. 629–643; Abstract: Eng. and Min. Jour. Vol. LXXXII, pp. 1066–1067.

1906. Contributions to Economic Geology, 1905; Investigation of Metalliferous Ores.
U. S. Geol. Survey, Bull. No. 285, pp. 14–19.

1907. Biographical Notice of George H. Eldridge.
Trans. Am. Inst. Min. Eng'rs. Vol. XXXVII, pp. 339–340.

1907. Uinta Mountains.
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U. S. Geol. Survey, Bull. No. 320. 75 pp.

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Abstract: Science, n. s. Vol. XXV, pp. 767–768.

1907. Investigations of Metalliferous Ores.
U. S. Geol. Survey, Bull. No 315, pp. 14–19.

1907. Suggestions for Field Observations of Ore Deposits.
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1910. The Cobalt Mining District of Ontario.
Abstract: Science, n. s. Vol. XXXI, p. 517.

1910. Criteria of Downward Sulphide Enrichment (discussion).
Econ. Geology. Vol. V, No. 5, pp. 477–479.

The Fritz Engineering and the Coxe Mining Laboratories of Lehigh University.

BY JOSEPH DANIELS,* SOUTH BETHLEHEM, PA.

(San Francisco Meeting, October, 1911.)

I. The Fritz Engineering Laboratory.

The Fritz Engineering Laboratory was built under the direction of John Fritz, and presented by him to the University. A view of the building, looking east, is shown in Fig. 1. The building was started in 1909, and completed in 1910, although all of the equipment was not placed until later. Mr. Fritz gave his personal attention to the details of construction and equipment, and it was his custom to drive over every day to the University from his home in Bethlehem and spend some hours watching the work, offering suggestions, making changes, and planning new work. The result is a building and an equipment which embody his practical ideas.

The laboratory structure is of the steel mill-building type, of light-colored brick, 91 by 114 ft., of which a section is shown in Fig. 2 and the floor-plan in Fig. 3. The steel frame carries the roof and traveling crane-way. Ample light has been provided by numerous windows in the side and end walls, in the clerestory, and by a skylight 84 ft. long and 9 ft. wide in the north roof. The main aisle of the building is 49 ft. 2 in. between centers of crane-columns, and has a clear height of 40 ft. The remainder of the width is taken up by two side aisles, 18 ft. high.

The laboratory consists of four sections: (1) A general testing-section containing the testing-machinery, a small machine shop, and an office; (2) a cement-testing room; (3) a room for making and storing concrete test-specimens; (4) a hydraulic section.

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* Associate Professor of Mining Engineering.

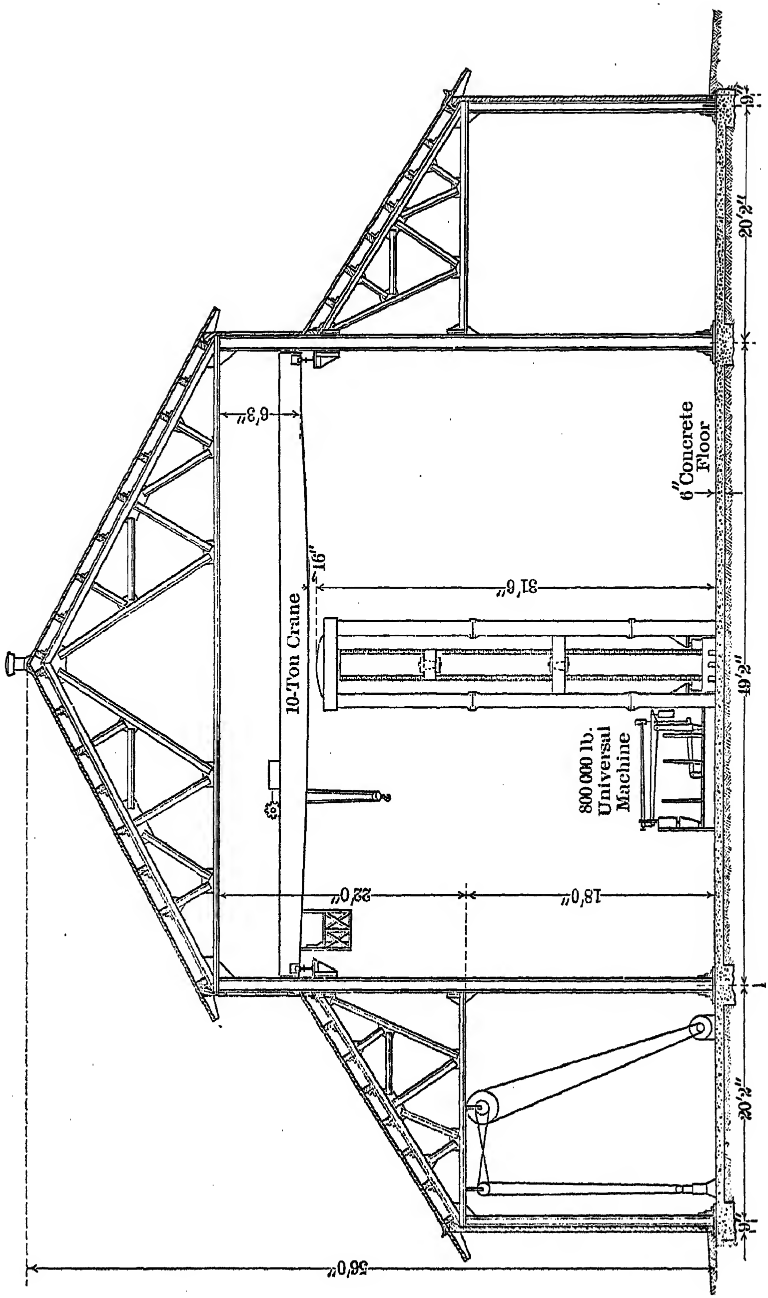

FIG. 2.—SECTION OF FRITZ ENGINEERING LABORATORY.

1. *The Testing-Section.*

This section, Figs. 4 and 5, occupies the larger part of the western end of the building and contains all of the testing-machines except the briquette-machines, which are in the cement section. For facility in handling the test-specimens, a 10-ton crane, 47 ft.

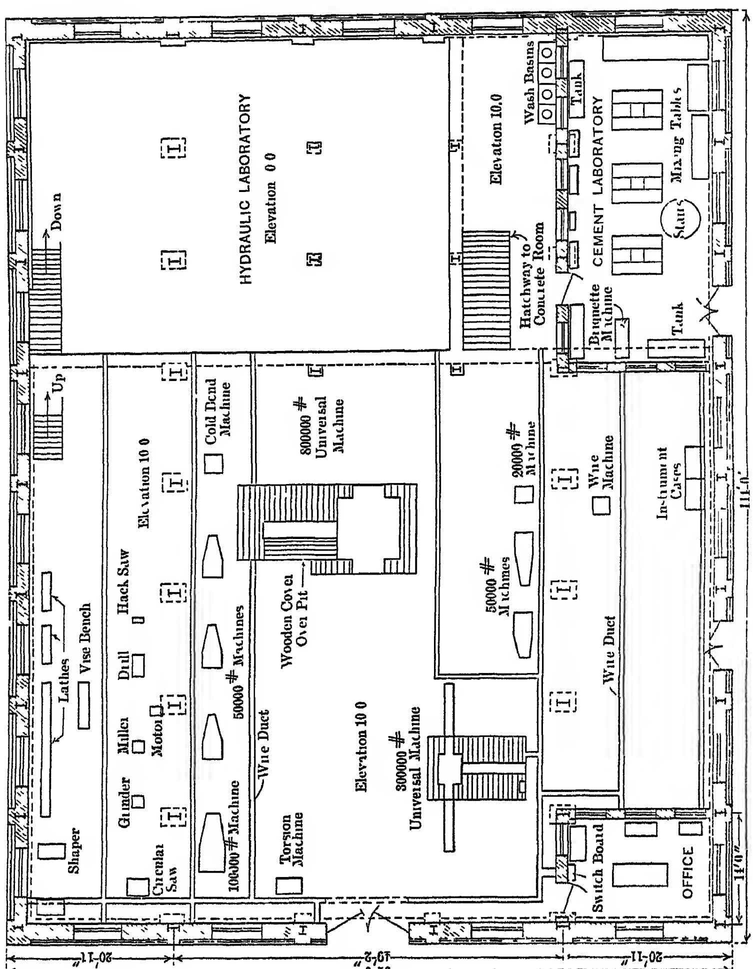

FIG. 3.—FLOOR-PLAN OF FRITZ ENGINEERING LABORATORY.

2 in. center to center of runway beams, operated by three direct-current motors, has been provided. A small machine-shop, containing a drill-press, lathe, milling-machine, shaper, etc., operated by a 7.5-h-p. motor, is available for general repair-work.

FIG. 1.—THE FRITZ ENGINEERING LABORATORY.

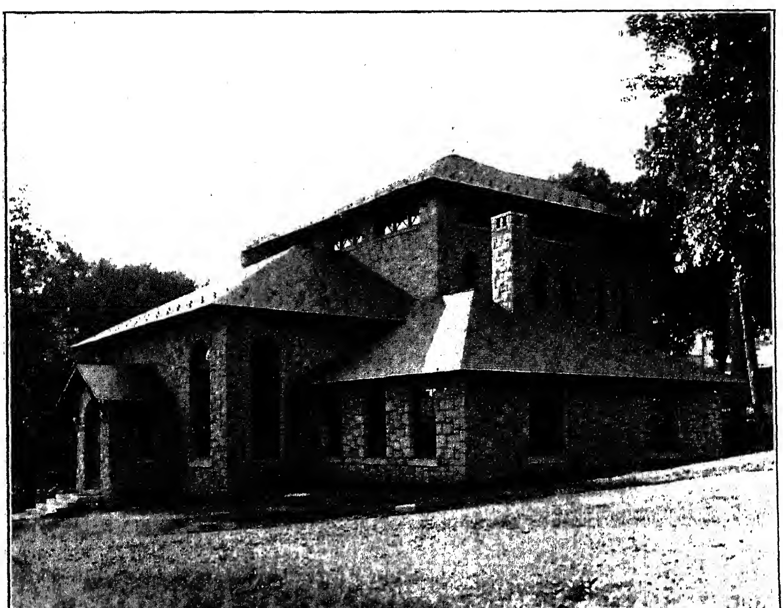

FIG. 6.—THE COXE MINING LABORATORY.

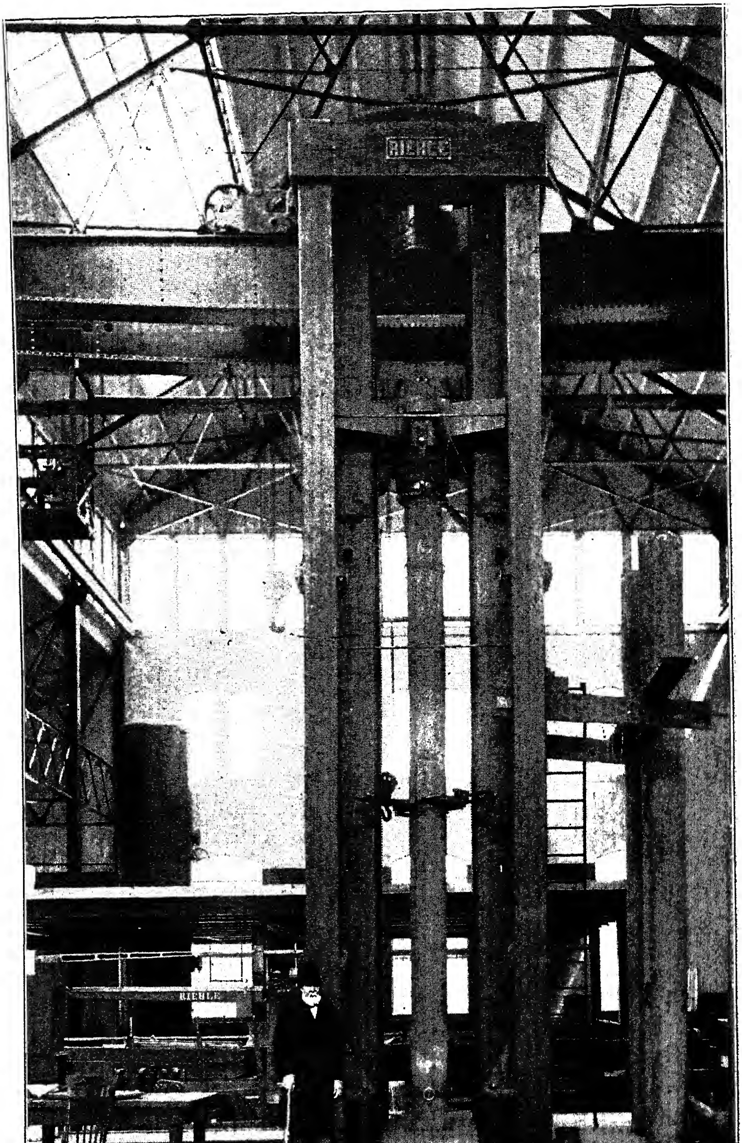

FIG. 4.—100,000-LB. RIEHLÉ TESTING-MACHINE, THE FRITZ ENGINEERING LABORATORY.

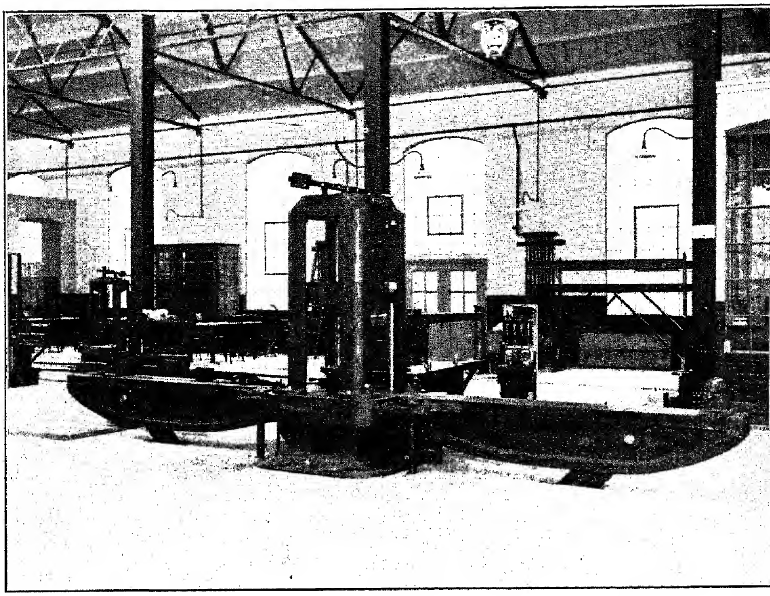

FIG. 5.—300,000-LB. OLSEN TESTING-MACHINE, THE FRITZ ENGINEERING LABORATORY.

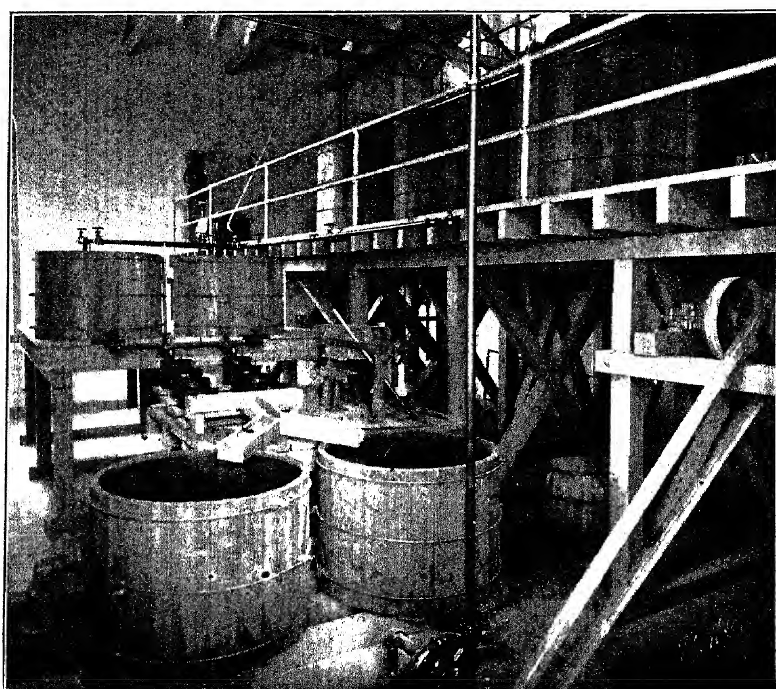

FIG. 7.—CYANIDE EQUIPMENT, ECKLEY B. COXE MINING LABORATORY.

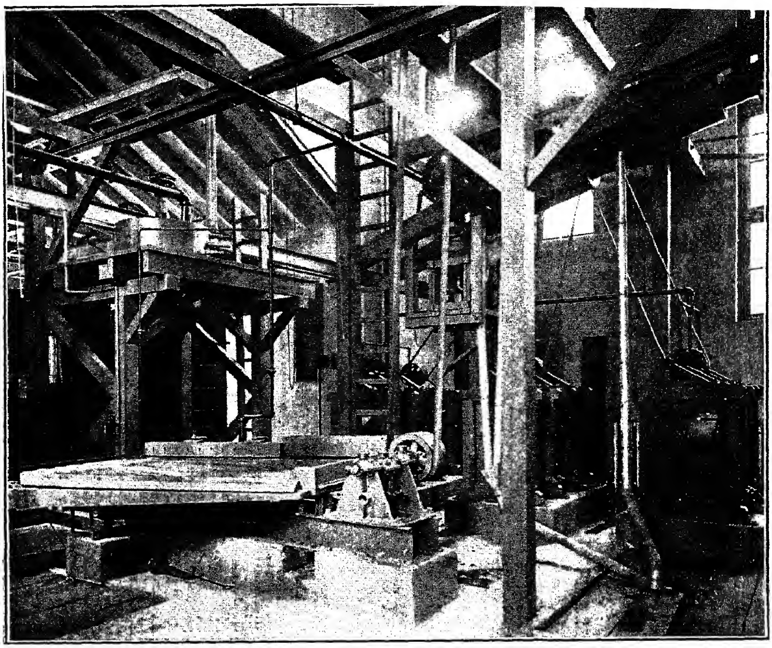

Fig. 10.—Jigs, Trommels, and Overstrom Table, Eckley B. Coxe Mining Laboratory.

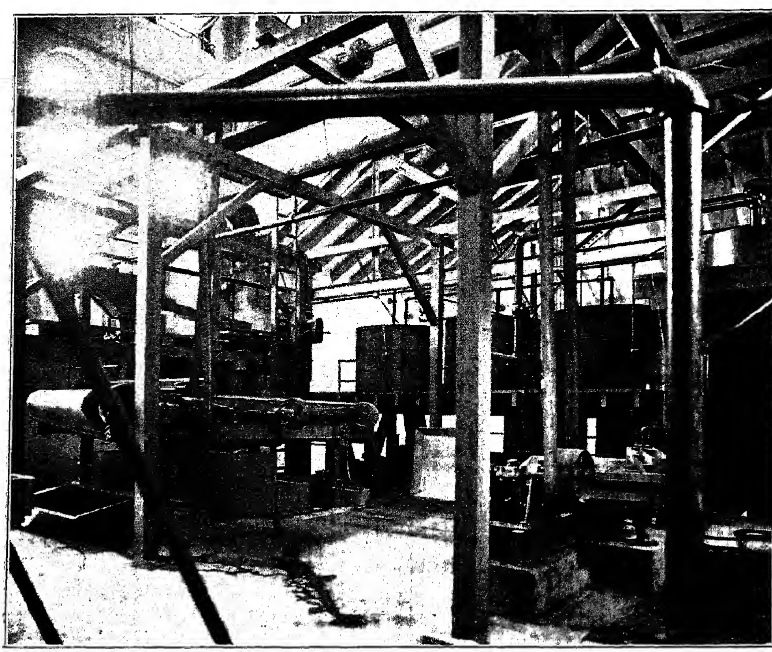

Fig. 11.—Classifiers and Tables, Eckley B. Coxe Mining Laboratory.

The principal equipment of the testing-section proper is as follows:

Type of Machine.	Capacity in Pounds.
Universal,	800,000
Universal,	300,000
Universal,	100,000
Universal,	50,000
Universal,	50,000
Universal,	50,000
Universal,	50,000
Universal,	50,000
Tension and compression,	20,000
Wire tester,	20,000
Cold bend, 1.5 in. diameter bar.	
Torsion, 24,000 inch-pounds.	

2. *The Cement-Testing Room.*

The cement-testing section occupies a separate room on the main floor. The equipment consists of tables for making cement specimens, storage-tanks, briquette-testing machines, and apparatus for making standard cement-tests.

3. *The Concrete-Room.*

The concrete-room is under the cement-room, and is used for preparing cubes, beams, and cylinders. It is connected with the main testing-room by a hatchway, through which the heavy specimens may be hoisted into the main room by the crane. The equipment consists of bins for sand and stone, mixer, and molds.

4. *The Hydraulic Section.*

The hydraulic section occupies the NE. portion of the building. The lower floor is 10 ft. below, and the second floor, or elevated platform, 10 ft. above the testing-room level.

The equipment on the lower floor consists of

1 DeLaval centrifugal pump, 2,000 gal. per min. against 60 ft. head.
1 Atlantic Hydraulic Machinery Co. centrifugal pump, 200 gal. per min. against 255 ft. head.
1 steel pressure-tank—65.25 in. in diameter by 34 ft. 6 in. high.
2 steel calibrating-tanks—8 ft. in diameter by 12 ft. high.
3 steel weighing-tanks—4 ft. in diameter by 3 ft. high.
1 steel Weir tank—4 by 4 by 21 ft. long.
1 Trump turbine.
1 Pelton water-wheel.
1 Rife hydraulic ram.

In addition, the upper platform carries

1 steel Weir tank—3 by 3 by 18 ft. long.
1 steel tank—6.5 ft. wide, 3 ft. deep, 17.5 ft. long.

This equipment also includes pressure-, mercury-, oil-, and hook-gauges, meters, scales, and so on.

All electricity for lighting and for power for the testing-machines and for the pumps is 2-phase, 60-cycle, alternating current at 110 and 220 volts.

The Fritz Laboratory forms part of the equipment of the Department of Civil Engineering, in charge of Prof. F. P. McKibben, to whom I am indebted for most of the preceding data. Instruction in testing and hydraulics is given to students in Civil, Mechanical, Mining, Metallurgical, Chemical, and Electrical Engineering, and Electrometallurgy in the junior year. The equipment is also available for thesis-work in the senior year, and for commercial tests on materials of construction.

II. The Eckley B. Coxe Mining Laboratory.

The Coxe Mining Laboratory is the gift of a friend of Lehigh, and was so named by the trustees in honor of Eckley B. Coxe, at one time an honored President of the American Institute of Mining Engineers, and during his life a devoted friend and trustee of Lehigh University.

Ground was broken for the building in October, 1909, and erection of machinery and equipment was begun in July, 1910. The main part of the equipment was ready for operation in the spring of 1911.

The building, designed by Furness, Evans & Co., of Philadelphia, the architects of Drown Memorial Hall, is of dressed sandstone in broken range style, steel roof-trusses, and finished inside with light Kittanning brick, as shown in Fig. 6. Its principal dimensions are 100 by 75 ft., one story high in front, and two stories high in back in the main part of the laboratory. The main or central part of the building contains the milling laboratory, 40 by 70 ft., built on two floors to secure proper fall for the machines; the two wings, one east and one west, are each 30 by 40 ft. The east wing contains a recitation-room large enough for 40 students, the department office

and library, an instrument-room for mine-surveying outfit; the basement contains a locker- and wash-room. The west wing contains room for a small ore-testing laboratory-equipment, such as screens, classifiers, tables, etc.; a chemical laboratory, and an assaying-room.

The laboratory is well lighted by windows extending the full height of the walls. In the milling laboratory, in the main walls under the eaves, sash-windows, operated from the floor by gearing and chains, furnish ventilation. In the wings, the direct-indirect system of heating and ventilation is employed. Flaming-arc lamps furnish artificial light, and individual lamp-sockets are provided for the various machines. Steam is used for heating and gas for auxiliary purposes.

The water-supply comes from the town mains, and is so arranged that it can be fed to a pressure-tank of 2,000 gal. capacity, or used directly from the mains. Drainage is by pump-pits and open floor-drains, all connected to a system of piping which discharges into a small creek near the building.

The framing and machinery have been painted a uniform light gray color. The concrete floors and pits have been treated with water-proofing paint.

1. *The Milling Laboratory.*

This section occupies the two floors of the main part of the building. The difference in elevation of these floors is 8.5 ft., the two floors being connected by steps and by ladders on the framing. The heavy crushing-machinery, stamp-battery, jigs, tables—all on the upper floor—are erected on substantial concrete foundations which extend nearly to the level of the lower floor. The upper floor, of reinforced concrete, is interrupted by the elevator and pump-pits. The ore-bins, feeding-platforms for the breaker, stamp-battery and grinding-pan, housings for elevators and screens, classifiers and settling-tank supports, and the supports for the motors and shafting, are all of yellow pine, framed construction. Ladders and floor-planks at convenient distances make the entire framework easily accessible.

On the lower floor of the milling laboratory is the cyaniding-department. The tanks, zinc-boxes, filter-press, and the agitation-pump are all carried on framing which extends in lifts up

to the level of the upper floor, thus getting a free fall to the sump-tanks and circulating-pump placed on the lower floor-level, Fig. 7. Typical mill-arrangement and construction, as far as practicable in a mining-school laboratory, have been followed.

The present equipment of the laboratory consists of the following machinery purchased from the Allis-Chalmers Co., and arranged as shown in Figs. 8 and 9:

1 grizzly, 2 by 4 ft.
1 Gates breaker.
2 vertical elevators, 6 in.
1 rolls and wall feeder, 18 by 10 in.
1 set of 3 trommels, 16 by 24 in.
3 3-compartment Harz jigs, 9 by 17 in.
1 Brown conical classifier.
1 Richards 1-spigot classifier.
1 Callow settling-tank, 4 ft.
1 Huntington mill, 3.5 ft.
1 Challenge feeder.
1 3-stamp battery, 500 lb.
1 Frue vanner, 4 ft.
1 Overstrom table, 7 ft.
1 grinding-pan, 36 in.
3 Frenier pumps, 6 by 48 in.
2 centrifugal circulating-pumps, 1.5 in.
1 water-tank, 2,000 gal.
3 solution-tanks, 5 by 4 ft.
3 leaching-tanks, 5 by 4 ft.
2 agitation-tanks, 6.5 by 5 ft.
4 gold- and sump-tanks, 4 by 3 ft.
1 filter-press.
Zinc-boxes, etc.

The machinery mentioned above is supplemented by all necessary fittings, chutes, pipes, trolley-crawls, blocks, and the like.

The electrical equipment consists of five induction-motors, 2-phase, 220-volt, 60-cycle, with auto-starters—giving a total of 50 h-p. Current is obtained from the University Power-Station at 2,200 volts, and is stepped down to 110 and 220 volts for lighting and power purposes.

The general plan and equipment was intended to show by actual example the more important types of ore-dressing machinery, and to give a means of demonstrating, by actual runs, the common methods of concentrating and treating the ores of

gold, silver, copper, lead, and zinc, by methods of coarse or fine concentration, amalgamation, or cyaniding. The arrangement of machinery, driven by five separate motors, permits tests to be made on one or a group of machines, or complete tests as a mill-run.

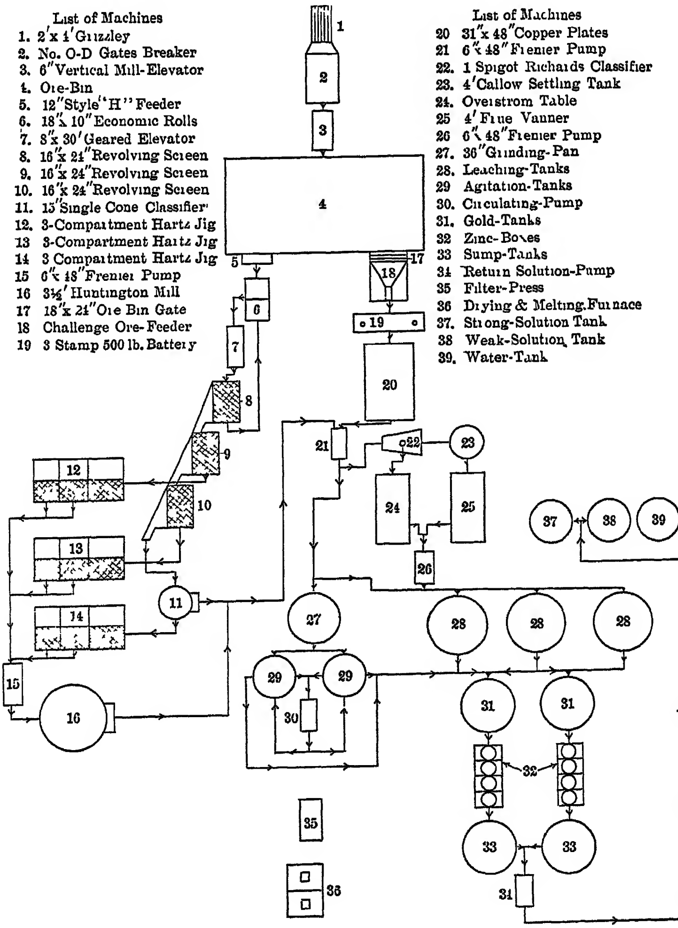

FIG. 8.—ORE-DRESSING EQUIPMENT, ECKLEY B. COXE MINING LABORATORY.

Ore is delivered at the rear end of the laboratory, where it may be fed direct to an elevator discharging to the bins, or it may be passed over the grizzly, through the Gates breaker, and then to the elevator. The bin is divided into two parts, one for base-metal ores and one for gold-ores.

On the coarse-concentrating side, Fig. 10, the ore is delivered by a wall feeder to the rolls, then to the elevator, and to the trommels and Brown classifier. This material may be jigged, or sent by a Frenier pump to the tables, or delivered to the Huntington mill for further reduction. The three jigs are three-compartment, Harz type. The crushing-machinery, elevators, jigs, and pump are run by a 30-h-p. motor.

The gold-ores are delivered to a Challenge feeder; then fed to the stamp-battery and plates. A 5-h-p. motor runs the stamp-battery. An amalgam-trap will permit the pulp to pass

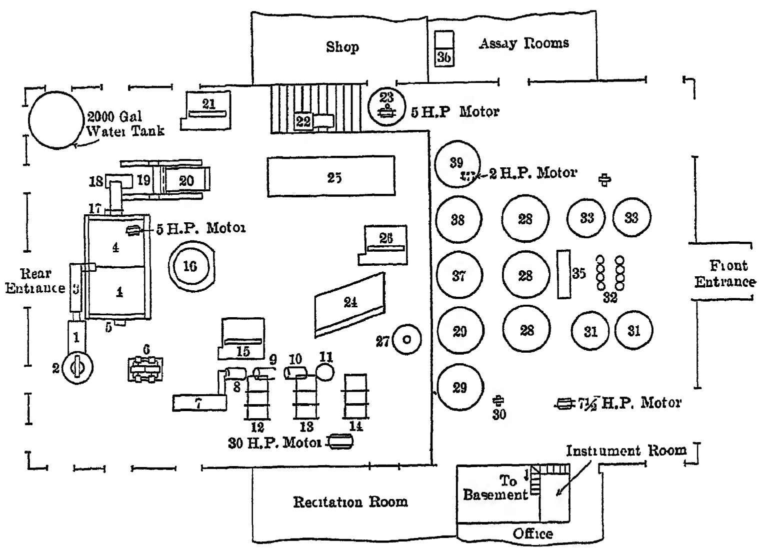

FIG. 9.—FLOOR-PLAN, ECKLEY B. COXE MINING LABORATORY.

to a second Frenier pump, which delivers its material to a Richards classifier and Callow settler. The classified products may be fed to either the Frue vanner, Overstrom table, or both, Fig. 11. Material from these machines is dropped to the third Frenier pump and sent either to the grinding-pan or directly to the agitation- or leaching-tanks. These latter machines are operated by a 5-h-p. motor.

The cyanide-plant consists of three solution-tanks, three leaching-tanks, two agitation-tanks connected by a 1.5-in. centrifugal circulating-pump, two gold-tanks, eight zinc-boxes, one filter-press, two sump-tanks, and a second 1.5-in. centri-

fugal pump to return the solution to the upper tanks. A 2-h-p. motor runs this pump; the other, together with the grinding-pan, is run by a 7.5-h-p. motor. The tanks are all of California redwood.

Ore-Testing Laboratory.

This part of the laboratory, not yet equipped, will eventually contain small crushers, rolls, and screens for the reduction and sizing of small batches of ore; laboratory-classifiers of Richards and Munroe types; hand-jigs, and small tables, together with all accessory apparatus.

Assay Laboratory.

This space is divided into three parts, one for fire-assays, one for wet-assays, and one for a balance-room. The instruction in assaying at Lehigh is in charge of the Chemistry Department, hence the room and equipment is intended only to handle the products of the laboratory. The usual outfit for assaying will be found here.

Library, Museum, etc.

The general library of the University contains most of the general books on mining, but in the department-office there is a small reference-library containing most of the books ordinarily required by students, all of the mining journals, and an excellent collection of catalogues, photographs, and blue-prints of mining-machinery.

The department also has a collection of air-drills, coal-cutting machinery, prospecting-drill, tipple-equipment, steel-timbering, mine-lamps, and the like, part of which is housed in the new building.

Scope of Laboratory.

The purpose of the laboratory is to familiarize the students with methods and practice of ore-treatment, and to develop a spirit of investigation and research. Instruction, at the present time, is given to the students of mining and metallurgy during the junior year, and will be extended to include the senior year. Lehigh has commonly been regarded as a coal-mining school; but the present equipment places it among those schools which also emphasize the metal side of the mining industry.

The Newport Iron-Mine.

BY B. W. VALLAT, IRONWOOD, MICH.

(San Francisco Meeting, October, 1911.)

THE Newport mine, located at Ironwood, Gogebic county, Mich., on the Gogebic iron-range, is owned and operated by the Newport Mining Co., for the mining of iron-ore.

I. GEOLOGY.

The general geology of the Gogebic range has been determined and recorded at different times by well-known geologists,[1] and their work will be referred to briefly in connection with the local conditions in the mine. The strike of the formation across this property is about 15° north of east. The general dip is 68° to the north. At the base of the formation on the south lies the granite. Looking north, at right angles to the strike of the formation, we have the foot-wall of quartz-slates and quartzite about 400 ft. wide; the iron-formation about 800 ft. wide, and the hanging-wall of black slates. The iron-formation is composed of banded jasper and quartzite together with iron oxide and concentrations of iron-ore. In character the ore is a soft red hematite, with occasional masses of hard blue "steel ore." The formation is crossed in many places by diorite dikes, which cut through at various angles, the larger or main dikes dipping into the foot-wall in a SE. direction at angles varying from 15° to 30°, while numerous small dikes occur which strike through the formation in a vertical plane and at approximately right angles to the main dikes.

There are various theories concerning these dikes, especially as to the time of their origin with relation to the ore-deposition. The commonly-accepted theory is that the dikes were there first, but there are some who contend that they were formed

[1] Irving and Van Hise, Penokee Iron Bearing Series of Michigan, *Monograph XIX., U. S. Geological Survey* (1892). C. K. Leith, A Summary of Lake Superior Geology, with Special Reference to Recent Studies of the Iron-Bearing Series, *Trans.*, xxxvi., 101 to 153 (1906).

subsequent to the deposition of the ore. Whatever their origin, they are there to-day, and in relation to the ore-bodies are of the greatest importance and must be carefully located and recorded in the mine-development. It may be said that these dikes have been responsible for a prolonged and serious interruption in the development of the Gogebic range. Assuming that the dikes were in place first, the most reasonable theory seems to be that of deposition by downward-percolating surface-waters carrying the original iron carbonates in solution, the iron oxide being precipitated by heat or other agencies as these waters became cut off, or ponded, in their downward course by impervious strata in the formation and deflected along the lines of least resistance. We know that the diorite dikes and foot-wall slates are impervious to water; that what we term our "main dikes" dip into the foot-wall at right angles, forming V-shaped troughs with the foot-wall, and a natural basin into which the iron-bearing waters would flow. Here they find their lateral limits and are backed up, or ponded, so that the process of precipitation and concentration is allowed to take place within the trough. The ore in the Newport mine, as so far demonstrated by the developments, is found deposited on a succession of these dikes underlying one another. It has also been found that there is a fault through the dikes, about 100 ft. north of the foot-wall, showing a throw of about 450 ft. east and west parallel to the foot-wall. This fault, found in the upper levels of the old mine some time ago, has been definitely located in the main dike on which the ore is now being mined. Since this faulting is general through the dikes so far encountered, it offers one way for the mineral-bearing solutions to get through from one to the other; and that each dike carries its own local deposit might be due to the fact that the waters of deposition flowing into these eastward-pitching troughs and finding their eastern limits, backed up towards the west and found their way through the fault-breaks in the nature of an overflow on to the next succeeding dikes.

II. History.

At the time the present owner purchased this property the mining was confined to an ore-deposit lying on a thick dike at a depth of about 600 ft. Some diamond-drill holes had been

put down into the formation below this dike on the Newport property, but with no encouraging results. Following the policy of developing a mine-operation well ahead of the winning of the ore, it was soon evident that sinking would have to be undertaken, and shaft *A*, the then deepest one, was selected for this purpose. The work was started in 1898. At a depth of 1,000 ft., or the 9th level, a small ore-body was found resting upon another dike. The shaft was continued through the dike and exposed only barren formation under it. From that time the work was carried on in the face of great difficulties and most discouraging conditions. Heavy water-flows were encountered, and the shaft drowned-out repeatedly, causing delays of days and weeks at a time in its progress. Moreover, it was difficult to keep the shaft open in places, due to the treacherous and broken-up character of the formation. With no encouragement in sight, and with the heavy financial drain necessarily attached to such conditions, it was persistent determination and effort, to say the least, that carried this shaft down a further distance of 750 ft. below the dike at the 1,000-ft. level, and penetrated the new ore-body, which opened the way to the subsequent development of this mine. In the year 1904, or six years from the time the work was started at the 7th level, the shaft struck into ore at a total depth of 1,800 ft. A theory, or opinion, which has been altered by this work is that the ores in this district would deteriorate in quality and become lean at a depth of from 1,000 to 2,000 ft. This opinion is still held by many who perhaps are not acquainted with the later developments in the district. Recent development of the two bottom levels of the Newport mine has shown up as high grade, clean, and concentrated a body of ore as any of the levels above, and this at a depth of nearly 2,400 feet.

III. Equipment.

The mine is equipped with a modern surface-plant, and new equipment of latest design is being added wherever it will increase the efficiency of the operation. The more important units only will be briefly mentioned in order to give a clearer idea of the general operation of the mine. The boiler-plant consists of six Wickes vertical water-tube boilers, five 250 h-p. and one 400 h-p., giving a total boiler-capacity of 1,650 h-p.

Each boiler is equipped with Roney stokers. The coal is handled into the boiler-house bins from the trestle stock-pile by means of belt-conveyors and a bucket-elevator. The engine-house, of brick-and-steel construction, covers an area of 56 by 163 ft. The plant in this building consists of two Thompson-Greer hoisting-engines of the Corliss type, with 24- by 48-in. cylinders, each hoist equipped with two tandem circular drums, 8 ft. in diameter by 12 ft. face, each drum containing individual friction-clutch and brake-gear for the purpose of hoisting in balance from any level. One hoist is used for ore exclusively, while the other operates the cages for handling men, timber, and supplies. Means are provided at the shaft for interchanging the cages for skips so that both hoists may operate four skips on ore at the same time, if desired. The remainder of the plant consists of a Nordberg cross-compound, two-stage, air-compressor, of 2,500 cu. ft. capacity, cylinders 16 in. by 32 in. by 42 in. steam, and 17.5 in. by 29 in. by 42 in. air; a Westinghouse 150-kw., 250-volt, generator, direct-connected to a Nordberg tandem-compound Corliss engine, 10 in. by 20 in. by 36 in.; a 250-kw., 250-volt, generator, driven by a 14-in. by 28-in. by 36-in. cross-compound Allis-Chalmers engine; and lastly, a 500-kw. mixed-pressure Curtis turbine equipped with an American regenerator and a Wheeler condenser. This equipment, which is a recent installation and somewhat new to a mine-operation, furnishes electric power for the entire operation, and replaces the reciprocating electric units, which are held in reserve.

It is evident that in the modern simple reciprocating-engines we obtain a very low percentage of steam-efficiency, especially in the hoisting-engine. For the purpose of utilizing the large amount of steam which is exhausted to the atmosphere, and converting it into useful power in the form of electric energy, the exhaust-steam from the hoisting-engines and compressor is conveyed to the turbine through the regenerator, which acts as the receiving-and-storage tank. Under normal hoisting-conditions there is enough exhaust steam to run the turbine at low pressure most of the time, thus generating, at a very small cost, electric power sufficient for the entire operation. When the low-pressure steam is insufficient to operate the turbine up to its required speed, due to inactive intervals between shifts

or delays in hoisting, high-pressure steam is automatically supplied to the high-pressure side of the turbine through a connection to the main steam-line for this purpose. This arrangement provides for the continuous operation of the turbine.

The machine-shop, a new brick-and-steel structure, is equipped throughout with individual motor-driven machines of the latest type. A new store-house, blacksmith-shop, carpenter-shop, laboratory, and hospital, of latest design and equipment, are now in course of construction.

The change-house or "dry" for the men, a brick-and-steel building of the latest design, is equipped for the cleanliness and comfort of the miners. It is a two-story or double-decked building, 32 by 187 ft. in plan; the floors are of concrete, graded so as to drain to a central gutter, which enables the keepers to flush the floors with a hose daily. The change-rooms are provided with shower-baths, stationary wash-basins, hot and cold water, and a set of two lockers for each pair of men, in order to provide for the safe keeping of the clean and working-clothes separately. The lockers are arranged in aisles with the open expanded-metal type for clean clothes on one side, and the sheet metal or inclosed lockers for working-clothes opposite, which latter are equipped with an expanded-metal bottom with hot-water heating-coils underneath, providing for a circulation of hot air through the locker to a hood at the top which leads into a pipe extending to the roof of the building. This allows the clothes to dry thoroughly between shifts, and at the same time conducts the foul air of the lockers out of the building. There are accommodations here for 768 men.

The above-described equipment pertains to *D* shaft, which is the main operating-shaft of the Newport mine. The shaft-house is of steel construction, with pockets and dumping-facilities for four skips. Self-dumping skips, each of 6 tons capacity, are used. The same general arrangement for handling the ore in the shaft-house is used in most of the mines in this and the other Lake Superior districts. A general view of the structure is shown in Fig. 1. In front of and connected to the shaft-house is a steel runway, Fig. 2, carrying a 5-ton electric crane for handling heavy timber, supplies, skips, and cages. A railroad-track extends under one end of the crane-runway, so that heavy material may be handled direct from the car to the shaft

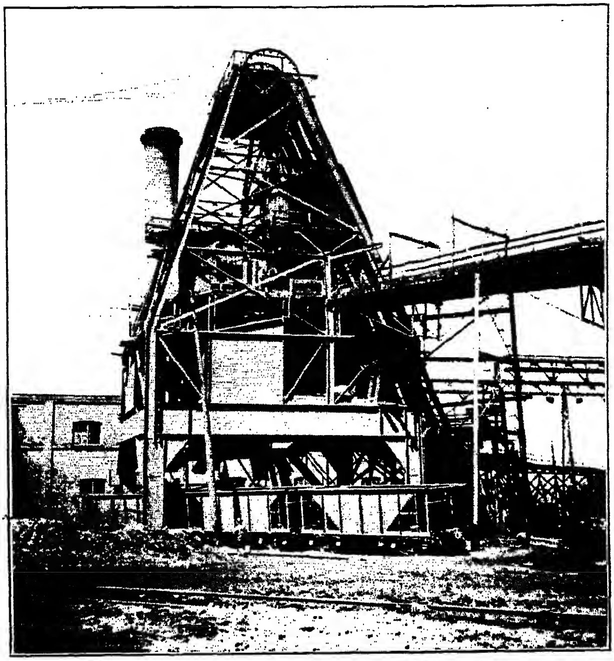

FIG. 1.—GENERAL VIEW OF *D* SHAFT.

FIG. 2.—*D* SHAFT CRANE.

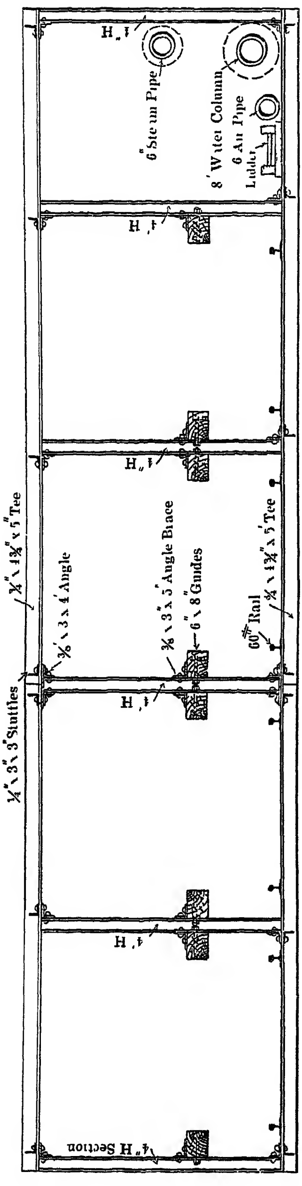

FIG. 3.—PLAN OF *D* SHAFT STEEL-WORK.

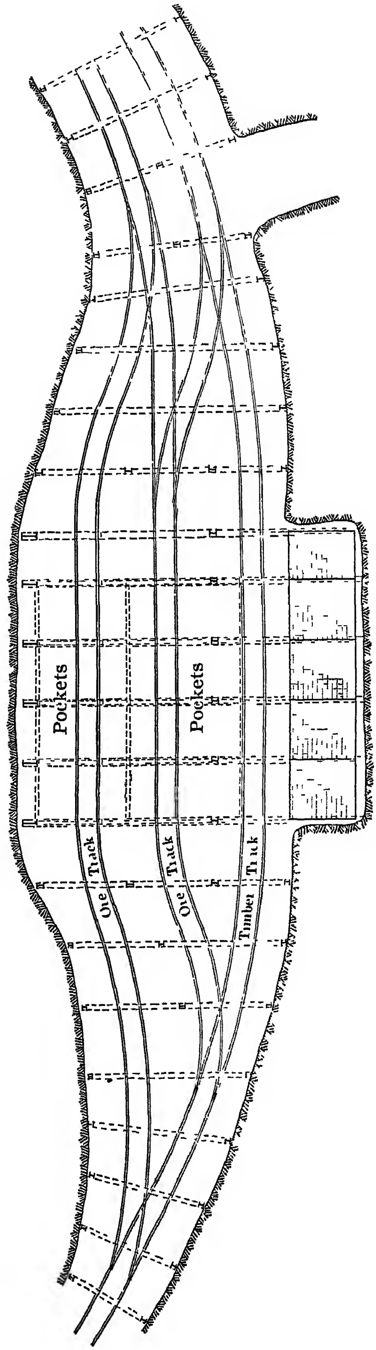

FIG. 4.—PLAN OF THE 19TH LEVEL STATION, *D* SHAFT.

very easily. This arrangement is very efficient and important, and is a great saver of time and labor.

IV. Mine-Equipment and Operation.

The operation of the mine for the past two years has been carried on almost entirely through shaft *D*. This shaft was put down in the foot-wall and started soon after the new ore-body was penetrated in the *A* shaft. It is lined with steel throughout and lagged with cedar lagging. The general plan of the shaft is shown in Fig. 3. The inside dimensions are 6 ft. by 28 ft. 6 in., divided into four hoisting-compartments, each 5 ft. 7 in. by 6 ft., and a ladder-and-pipe compartment 4 ft. 4 in. by 6 ft. It lies on the dip of the foot-wall, or at an angle of 68°, and is now 2,400 ft. deep, measured on this angle. For most of the time two of the hoisting-compartments are used for ore, and two for cages, all four being equipped with a 4-ft. gauge track laid with 60-lb. rails bolted direct to the wall-plates, and 6 in. by 8 in. tamarack back-runners or guides.

Before proceeding with a description of the underground equipment and the general operation of the mine, I wish to mention here the production made at this shaft during the year 1910. The total production for the year of 307 working-days was 1,074,800 tons, or an average of 3,500 tons per day. The best daily hoist was 6,652 tons in 21 hr., and the best month, 112,719 tons, in August. The extreme hoisting-distance was 2,400 ft., and the average about 2,150 ft., the production coming from four different levels. The maximum hoisting-speed was 2,200 ft. per minute. Five separate grades of ore were maintained and shipped, and the handling of the men, timber, and supplies necessary for the operation was also done in this shaft. So far as I know, this is the record tonnage-production in this country for a single, deep-mine shaft. It must not be construed that this production was made for record purposes; on the contrary, it was the natural outcome of a heavy year's requirements for delivery which made an expeditious operation imperative. It will, therefore, be of interest to know something about the equipment, method of handling, and system of mining, which made this production possible from one shaft.

The main-level stations in the mine, which are established in front of the shaft, are equipped with pockets and slides for receiving the ore from the mine-cars and loading it into the skips for hoisting. These stations and pockets are of steel construction, shown in Figs. 4 and 5 of the 19th level station.

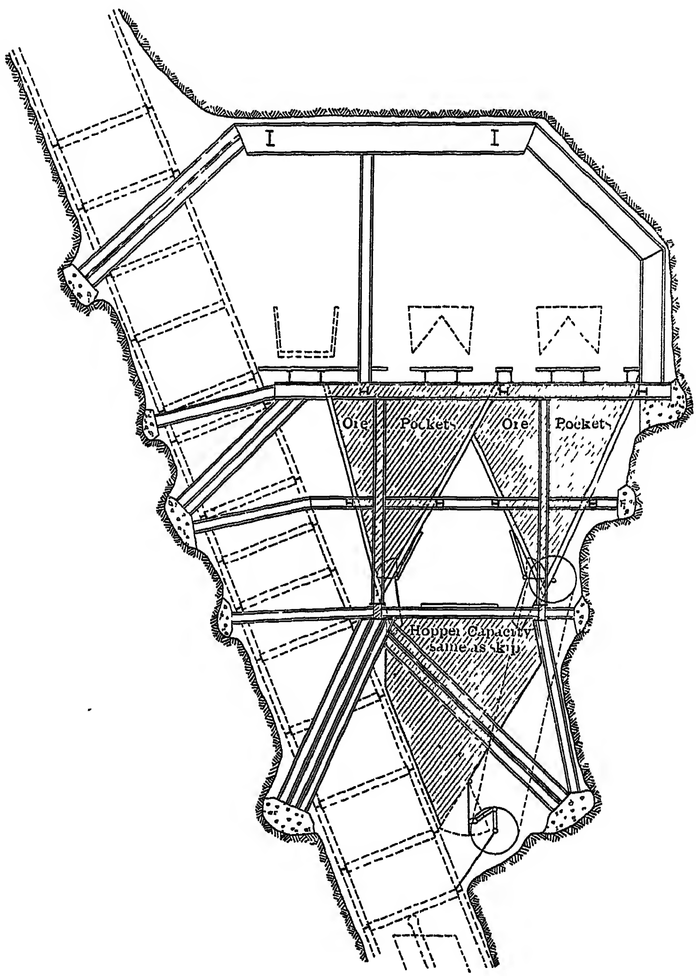

FIG. 5.—STATION AND POCKETS AT 19TH LEVEL, *D* SHAFT.

There are eight receiving-pockets under the floor of the station, which take care of the different grades of ore and serve four skips, when necessary. The total storage-capacity of the eight pockets is 200 tons of ore. Two parallel tracks, laid lengthwise

across the station, are connected to the track-system of the level. Each track serves a set of four pockets, thus giving a pair of pockets for each skip-road. A third track is brought in across the station on the 19th level, next to the shaft, for timber- and supply-trucks, which can be spotted in front of the cages to receive their loads without blocking the ore-traffic. Below each pair of shaft-pockets, and directly under the discharging-chutes, is a secondary pocket, or measuring-slide, for each skip-road. These slides hold approximately 6 tons, or a full skip-load of ore, and are filled between trips; that is, while a loaded skip is being hoisted to the surface and an empty one is returning, a load is "measured out" from the main pocket chute above and is ready, so that the instant the returning empty skip touches the shaft-gate, on which the skip rests when receiving a load, the stop or door of the slide is thrown open by the skip-tender and a full load dumped into the skip as fast as gravity can take it. This takes place while the loaded skip, which was hoisted, is dumping into the shaft-pockets on surface. The time required for this operation is about 4 sec., which is the interval between trips when the hoist is at rest. This arrangement is one of the main features which make rapid hoisting possible at this mine.

The plan of the 17th level, Fig. 6, shows the regular method of a main-level development of the ore-body from the shaft. As a general rule, the drifts and cross-cuts are driven 100 ft. apart where it is possible to do so, subject to variation due to the horses of rock which are occasionally encountered in the ore-body, the object being to avoid all development in rock except where absolutely necessary. All openings have to be well timbered. On the main levels 8-ft. sets (8-ft. posts and caps) of heavy timber are placed 6 ft. apart as the openings are being driven, and back and sides are closely lagged between sets. This work must be carefully done, as it is necessary to maintain these openings for several years, since a main level, after being opened up, becomes the operating-level for the transportation of the ore from the sub-level mining above, as will be shown later. It is obvious that the opening-up of the ore-body by main levels well ahead of the actual mining is necessary, and besides developing the future possibilities and future operation of the mine, serves a most important purpose in draining the

overlying ore of the water which usually accompanies a soft-ore deposit. It also tends to regulate the flow of water, making a fairly-uniform pumping-operation possible. In this ore-deposit the water drains off very rapidly to the bottom level, so when the actual mining takes place the ore is very dry and is much more easily handled. When an extra heavy flow of water is encountered in driving a new opening, the work is stopped and the ground allowed to drain until the flow diminishes. The water handled in the Newport mine is remarkably light for such a deep mine, not exceeding 350 gal. per minute.

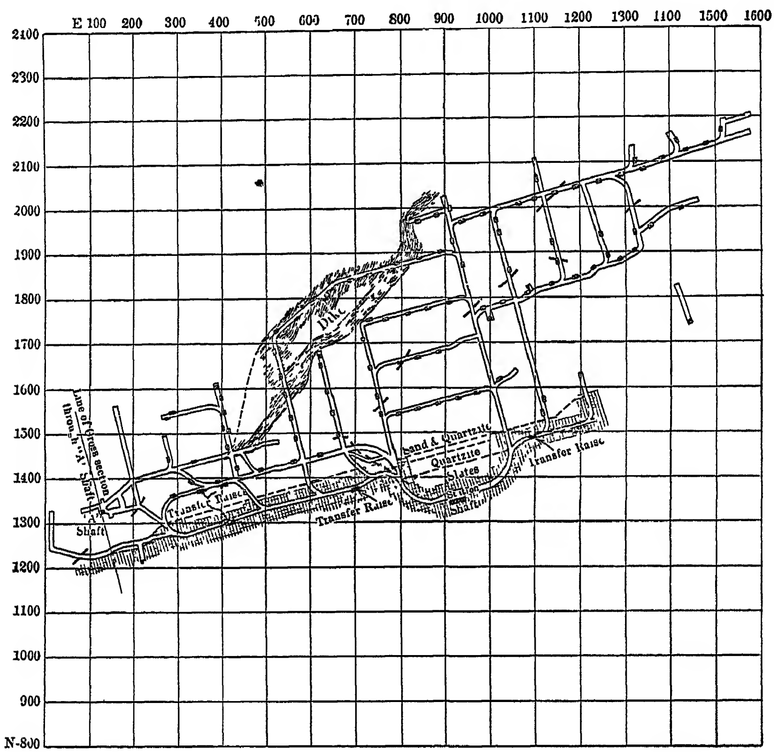

Fig. 6.—Plan of the 17th Level, *D* Shaft.

The system of mining used is the "sub-slicing" system, sometimes referred to as "top" slicing, also as "slicing and caving." Caving is necessarily a component part of this system. Referring to Fig. 6, it will be seen that raises are indicated along the drifts and cross-cuts about every 50 ft. These raises, 5 by 7 ft., and lined with cribbing, or 6-in. round timber, are put up

to the main level above, usually a vertical distance of from 75 to 100 feet. They are established wherever possible at a maximum distance of 50 ft. apart, in order to eliminate any long trams from working-places in the sub-levels. From these raises, sub-levels are opened up in their proper order by means of drifts and cross-cuts connecting the various raises, and later sub-divided into 50 ft. pillars, see Fig. 7, just before the final mining-out system begins. Sub-levels are opened out every 15 ft. between the main levels, with the exception of the first sub

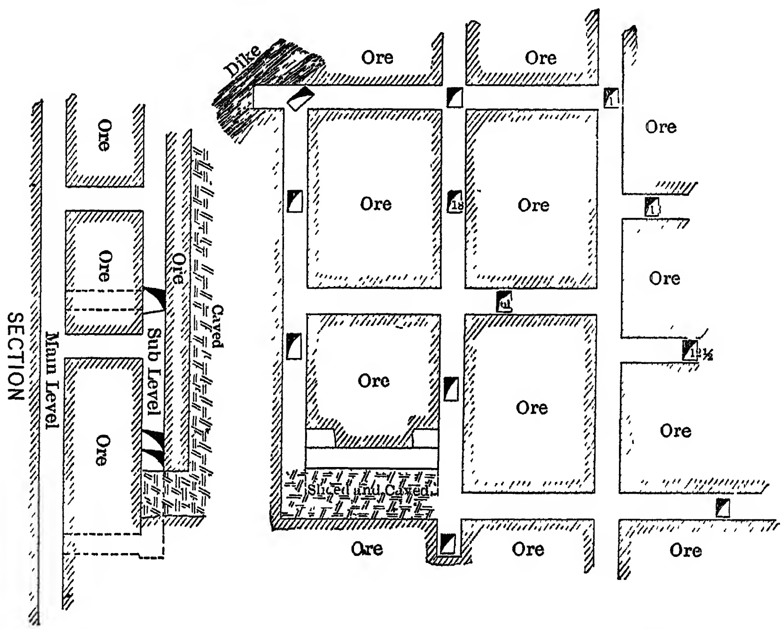

FIG. 7.—PLAN OF A PORTION OF A SUB-LEVEL, SHOWING METHOD OF BLOCKING OUT ORE AND LOCATION OF RAISES.

above a main level, which is established at a height of 18 ft. in order to allow 3 ft. more of a back over the main level for better protection, thus making slices of this thickness, which are mined out in blocks or sections from 300 to 400 ft. east and west along the ore-body, as shown in Fig. 8. In all the sub-level work, 7-ft. timber is used, and as the openings require only temporary support, smaller timber is used than on the main levels, usually from 8- to 12-in. round timber. The

ground is closely timbered and lagged, the sets being placed from 4 to 5 ft. apart, according to the nature of the ground.

To start this system of mining, a main level is opened somewhere near the top of the ore-body, within 50 or 75 ft., and raises put up to the rock capping. At an average of 15 ft. below the capping, a sub-level is opened out from the raises in the extreme eastern end of the ore-body back towards the west for a distance of 300 ft., thus making a first section 300 ft. long, and the full width of the ore-body from foot- to hanging-wall. The eastern extremity of this section will be immediately under the capping, owing to the eastward pitch, Figs.

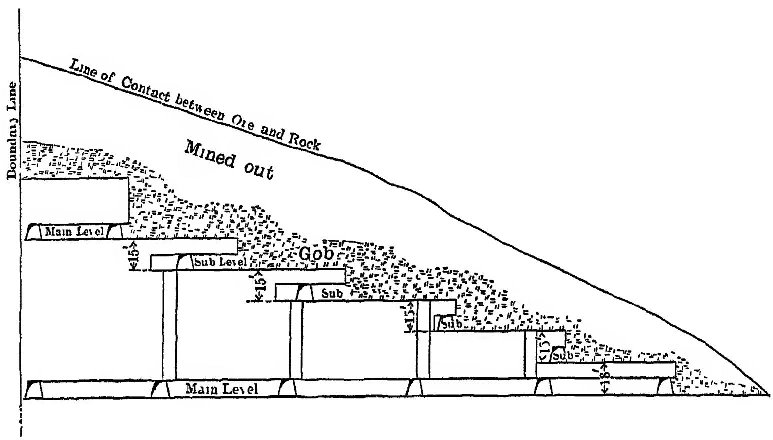

FIG. 8.—LONGITUDINAL SECTION OF MINING IN 300-FT. BLOCKS OR SECTIONS. VERTICAL SCALE EXAGGERATED.

8 and 9. This section is then split up into 50-ft. pillars, Fig. 7, and the final step in mining out the ore starts usually on the hanging-side of the ore-body. The first slicing-drift or cross-cut, *A*, Figs. 9 and 10, is now driven, the same size as the regular sub-level drifts, timbered and lagged in the same way, on the inside of the first pillar. Immediately alongside of this drift another one, *B*, is driven parallel to *A*, so that the adjacent legs or posts of the sets in each opening overlap one another. There now remains a slice of ore above *A* and *B* from 6 to 7 ft. thick. In Fig. 9 and in the third sketch of Fig. 10, the two final steps in the operation are shown together. The ore over *A* is blasted down by drilling short holes, to guard

against disturbing the rock capping and allow the ore to fall away from it clean. The timber sets are left to stand if they will. The ore is then shoveled into the small 0.5-ton sub cars or "buggies," trammed to the nearest raise and the ore dumped into it. All this is done by the men working under the protecting timbers of *B*. When all the ore over *A* is taken out the full length of the slice, the floor is "covered down" with old lagging, blocking, or pieces of timber, and the original timber sets, which are left behind to accumulate with it, Fig. 9. Upon this the unsupported rock capping keeps shelling off and dropping, so that this covering protects the next sub-slice

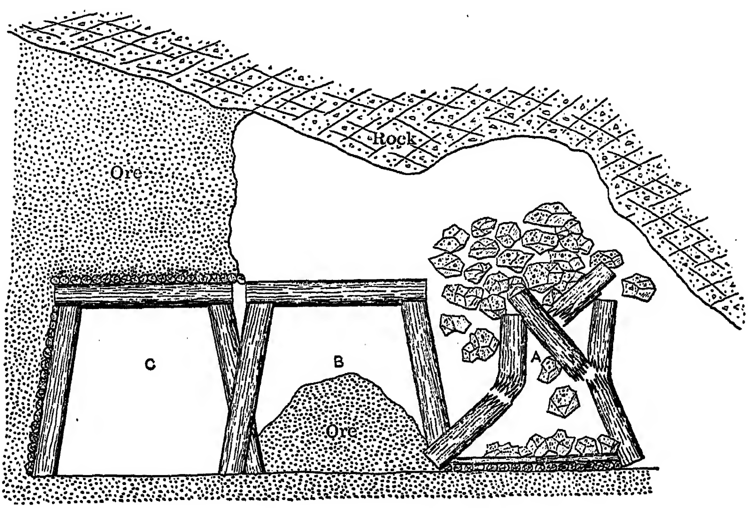

FIG. 9.—SLICES UNDER ROCK CAPPING.

directly underneath from the rock and sand mixing in with the ore when it is being mined out in its turn. A third drift slice, *C*, is then driven next to *B*, and this process repeated until the pillar, and eventually the entire sub-level, is all "pulled back" or mined out. When this is completed the entire floor of the mined-out section is covered, as stated above, with the rock capping left to cave down of its own weight on the timber covering. As the mining is carried down to the successive sub-levels below, the old timber is allowed to accumulate with the timber covering above as it slowly caves down, the entire mass forming what is called the "gob," Fig. 10. The gob

plays a very important part in this system of mining. As it gradually descends with the mining-out of the sub-levels, it grows larger and heavier, not only with the timber it accumulates but with the rock capping which is continually dropping down on top of it, until it is now a great immense network or

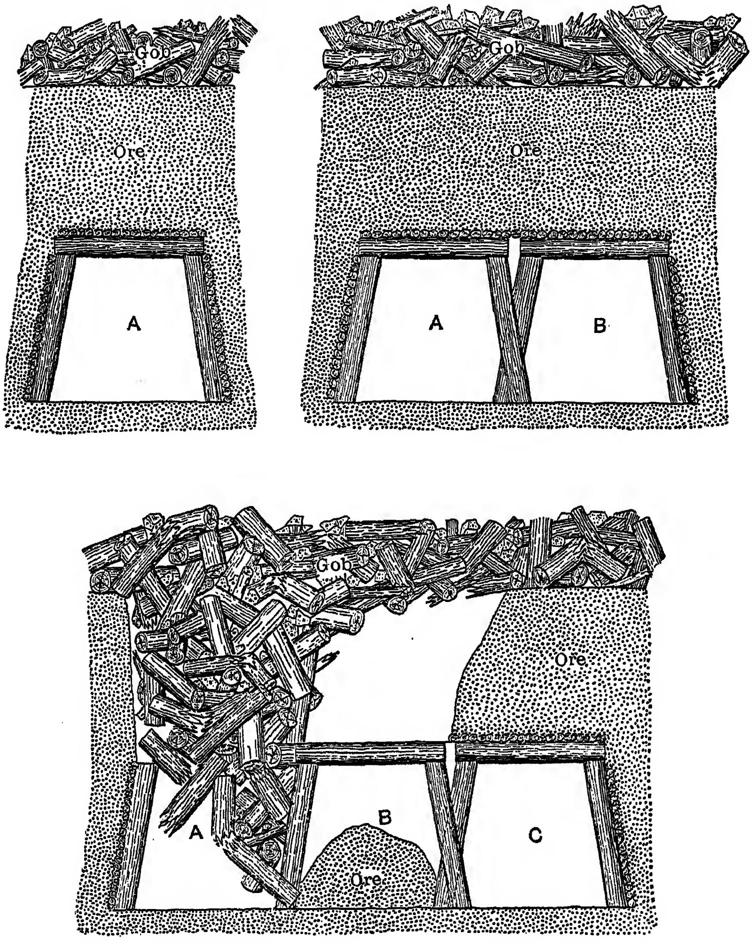

FIG. 10.—SKETCH SHOWING METHOD USED IN SUB-LEVEL MINING.

matted mass of timber, under tremendous pressure, slowly crushing down on top of the ore. It is very evident that this gob must form an ideal and perfectly safe roof under which the ore is mined out. The very nature of its make-up prevents it caving in suddenly, and while it is continually working or set-

tling down, it gives plenty of warning to the men by the creaking of the timbers, especially in spots where it tends to cave down faster than is usual. It also forms a cushion which effectually absorbs the shock of an extra heavy caving-in of the rock capping above.

While this first section, or block of 300 ft., is being mined out, the next section, 300 ft. west on this same level, is being opened and prepared as described above, so that by the time the first block is mined out the second section is ready for the same process. At the same time the first section on the next sub-level, most of which is directly under the rock capping, is being opened out and mined. In this way the uppermost sub-level is kept 300 ft. to the west in advance of the level next below, and so on down. This part of the system is strictly adhered to, and a sub-level section is never opened up for mining beyond the usual first few drifts and cross-cuts until the section directly above has been mined out. When a main level is reached as the mining progresses downward, it is treated exactly as a sub-level and mined out in the same way, the ore going down through the raises to the main level below. The amount of development of sub-levels ahead of actual mining depends altogether upon the production required. It is desirable to keep this development down to the minimum, and it is important that the ore be mined out as quickly as possible after development, as the timber can only be depended upon to hold up the ground temporarily at best. Where so many raises are available, they offer as many points of attack in opening up a new sub-level, which can be done in a very short time, if necessary. These raises also facilitate the grading of the ore which has to be separated. In general, this system is most satisfactory for mining this ore-body, and besides being safe for the men, in that they are always protected by timber and have safe openings behind them for retreat, it permits of a very clean and high percentage of ore-extraction. It is the policy in the operation of this mine to use a very liberal amount of timber in order to gain as nearly as possible a complete extraction of the ore, in addition to making safe working-conditions for the men. To give an idea of the timber used, it required about 653,500 lin. ft. of drift timber and 5,278 cords of lagging to mine the ore produced in 1910.

The ore is trammed out of the working-places to the nearest raise in the small buggies, which run on tracks laid with 8-lb. rails. Turn-sheets or iron plates are used at the intersection of the drifts, to turn the corners. The ore is then run out of the raises through chutes over the main-level tracks and into the cars which are spotted underneath. The main-level cars are of 2 tons capacity, and of the double side-door-dump pattern. The main-level tracks are laid with 30-lb. rails. The electric-haulage system, used throughout, is operated with 4.5-ton electric locomotives. The loaded cars, standing in groups of from 3 to 6 at the various chutes, are made up into a train of from 10 to 15 cars, hauled to the shaft-station and dumped into the pockets, according to the grade. Two men

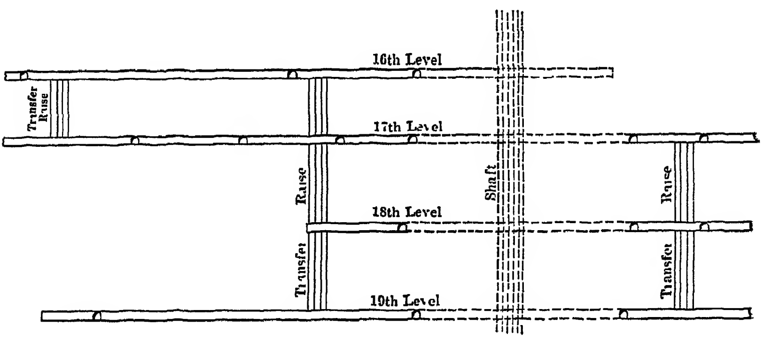

FIG. 11.—LONGITUDINAL SECTION THROUGH TRANSFER-RAISES.

stand on either side of the train as it comes into the station, and as each car passes over the proper pocket, the door-catches are tripped with hammers and the ore falls into the pockets, from which it is loaded into the skips as before described. A train of cars is very quickly dumped in this way as it passes over the pockets.

Fig. 7, at the 17th level, shows a long drift driven within the foot-wall rock and connected to the main cross-cuts in the ore-body, which provides a permanent haulage-way for the ore-traffic; and since the main haulage-ways in the ore-body are uncertain and liable at any time to block the traffic, due to the breaking-down of a set of timber, and will eventually be obliterated when the level is mined out, it makes a safe, sure and self-maintaining outlet. As a provision against delays or a tie-up of any kind at the main-level stations, a system of

transfer-raises in the foot-wall drifts has been arranged, which are really small shafts, 3 by 15 ft. inside the cribbing, and divided into three compartments, two for ore and one for a ladder-road, extending from one main level to another, as shown in Fig. 11, and indicated on the main-level plan. This system is operative from all the main levels down to the bottom, or 19th, level, and with the connecting foot-wall drift between *D* and *K* shafts, described below, the entire product may be transferred to and hoisted through *K* shaft. In the event of a tie-up at a main-level station which will stop hoisting at this point, the ore-trains dump into the transfer-raises instead of the station-pockets, and the ore is hauled to the shaft-station from the transfer-chutes on the next level below. If the delay is a short one the transfers serve as storage-raises, and the ore is allowed to remain in them until such time as it can be handled without interfering with the regular traffic on the level from which it is to be hoisted. If the hoisting-operation is delayed long enough to allow the transfer-raises and sub-level raises to become full, the ore may be conveyed to the bottom level through the transfers and finally over to *K* shaft, through the connecting-drift, to be hoisted. This makes the operation of handling the ore underground very flexible, so that it would require an unusual combination of circumstances to tie up the production completely.

Shaft *K*, situated 0.5 mile east of *D*, is the only other operating-shaft of the Newport mine, and while it has not been operated continuously on its own ore-bodies, it has been connected with *D* shaft on the bottom level by a long foot-wall drift, as noted above, thus serving as an excellent auxiliary outlet for *D* shaft, both for ore and men when necessary, and also greatly improving the ventilation of the entire mine. Connection is also maintained with shaft *A*, both for ventilation and as an emergency-outlet for the men. The connecting-drift between *D* and *K*, 2,600 ft. long, is a 10- by 10-ft. opening, all in the foot-wall rock, and is equipped with a track of 40-lb. rails, electric haulage, electric lighted. This drift was driven from both shafts at the same time and connected up, or "holed," in November, 1909, 140 days after starting the work. The best month's record was a total of 595 ft. driven by both parties in 26 days, 287 ft. from *D* and 308 ft. from *K*.

With the exception of a small amount of ore, shipped by rail during the winter months, the production is stock-piled on surface until the season of navigation opens on the Great Lakes. During the navigation period the shipments are made from the shaft, and from the stock-piles by means of steam-shovels. The ore is shipped by rail to Ashland, the nearest Lake Superior port, and into the ore-docks, from which boats are loaded for Lake Erie ports. During the year 1910 there was stocked 309,000 tons from *D* shaft during the winter months, and shipped during the season of navigation.

A recent installation of interest is a new pump-house, 30 by 60 by 18 ft., cut out in the solid granite back of the foot-wall on the bottom level, and connected to the main drift by a cross-cut. Installed here is a Prescott crank-and-fly-wheel, cross-compound Corliss pumping-engine, 22 in. by 42 in. by 4.75 in. by 36 in., with pot-form water end. This unit pumps direct to the surface against a vertical head of 2,150 ft., at a total capacity of 500 gal. per minute.

Notes on the Liberty Bell Mine.

BY CHARLES A. CHASE, DENVER, COLO.

(San Francisco Meeting, October, 1911.)

THIS paper, descriptive of a single mine, is presented in the belief that it may furnish useful suggestions to mine-managers encountering similar problems; and it includes the details which will enable them to estimate the value of the methods employed—especially where these depart from common practice. It should be added, however, that this mine is not typical of the San Juan district, but differs markedly from neighboring mines in physical conditions and metallurgical requirements.

The Liberty Bell mine is situated in San Miguel county, Colo., on the west front of the San Juan mountains, 2 miles north of Telluride. The vein was discovered in 1876 by Wm. L. Cornett, who, with subsequent locators, took up claims along the apex. A few hundred feet of development-work was accomplished, and a few tons of ore were smelted or milled; but profitable working proved impossible under existing

conditions; and the property lay idle until 1897, when Arthur Winslow acquired it for the United States & British Columbia Mining Co. After due investigation and preliminary development, and the initial construction of mine-buildings, tramway, and a ten-stamp section of the proposed 80-stamp mill, the Liberty Bell Gold Mining Co. was organized, and operations began in December, 1898. There have been, since that date, two complete suspensions, aggregating 10 months, for extensive additions and alterations at the mill; a suspension of 3 months in 1902 for reconstruction, following disastrous snow-slides; and one for 4 months in 1903, by reason of labor-troubles—a total of a year and a half. Otherwise, the working of the mine has been continuous, and production has expanded to 400 tons of ore daily.

The revival of the enterprise by Mr. Winslow took place at the time when the treatment of raw mill-tailings by direct cyanidation was first demonstrated as profitable, and within a year or two after the first successful long-distance transmission of electric power. Experiments in the cyaniding of tailings from amalgamation and concentration were begun almost immediately. In September, 1899, an experimental 7-ton leaching-plant was installed, under the direction of J. W. Mercer and F. L. Bosqui; and in May, 1900, a 250-ton leaching-plant of the South African type was ready for operation. It was evident from the outset that the mine could be made profitable, although this plant treated probably the lowest grade of material then handled in this country by this process. At the same time, the Telluride Power Co. had just undertaken to furnish power to customers; and the success of this pioneer enterprise helped to render possible the profitable operation of the Liberty Bell.

The Mine.

The vein occupies a strong fault-fissure, developed principally in the San Juan formation of andesitic flows, tuffs, and breccias, and striking almost exactly NW. and SE., though bearing more nearly E. and W. than the Tom Boy, Argentine, Smuggler-Union, and other veins of the immediate vicinity. The average dip is nearly 57° SW., but varies from 45° to vertical, being in general flatter and more irregular than that in neighboring productive mines. Though traceable on the sur-

face for 3 miles or more, the productive development is confined to 6,500 ft. The average width in the area worked for the past 12 years (including pinched and unworkable ground) has been 3 ft. In the ground actually stoped, the width has been 4.3 ft. In other words, 56 per cent. of the area opened has proved workable. These figures are based on the estimate of 13 cu. ft. as the bulk of a ton of ore in place, and on the area mined and the tonnage produced.

The vein-material is generally crushed, loose, and oxidized, indicating much movement in the plane of the vein. Hard massive quartz and calcite occur, but the more common appearance is that of more or less regularly banded quartz or calcite, or both, with bands or bunches of feldspathic country-rock, in various stages of alteration and silicification, and often completely kaolinized. This kaolin is a distinctive feature of the vein, occurring in masses, filling interstices, and as a gouge along one or both walls. Commonly brown or black (presumably by reason of the presence of manganese), it gives those colors to almost all ore produced. Rhodonite and rhodocrosite are occasionally seen.

The following analysis,[1] by Arthur Winslow (Column A), of Liberty Bell ore was made at the Massachusetts Institute of Technology in 1898. Column B indicates what the ore should be, unoxidized:

	A. Per Cent.	B Per Cent.
Quartz and clay (insolubles),	84.44	84.44
Calcite,	9.45	9.45
Apatite,	0.20	0.20
Limonite,	3.30	
Pyrite,	0.35	3.60
Arsenopyrite (?)	0.24	0.24
Bornite,	0.15	0.23
Undetermined (C, H_2O, and loss),	1.87	1.84
	100.00	100.00

The upper workings showed general oxidation, and when the Stilwell tunnel cut the vein, 1,000 ft. lower, the first evidence of proximity was black mud, gushing from the drill-holes. Subsequent development at the lower level and in the raise connecting to the upper workings has shown oxidation as complete as in the zone above.

[1] *Trans.*, xxix., 293 (1899).

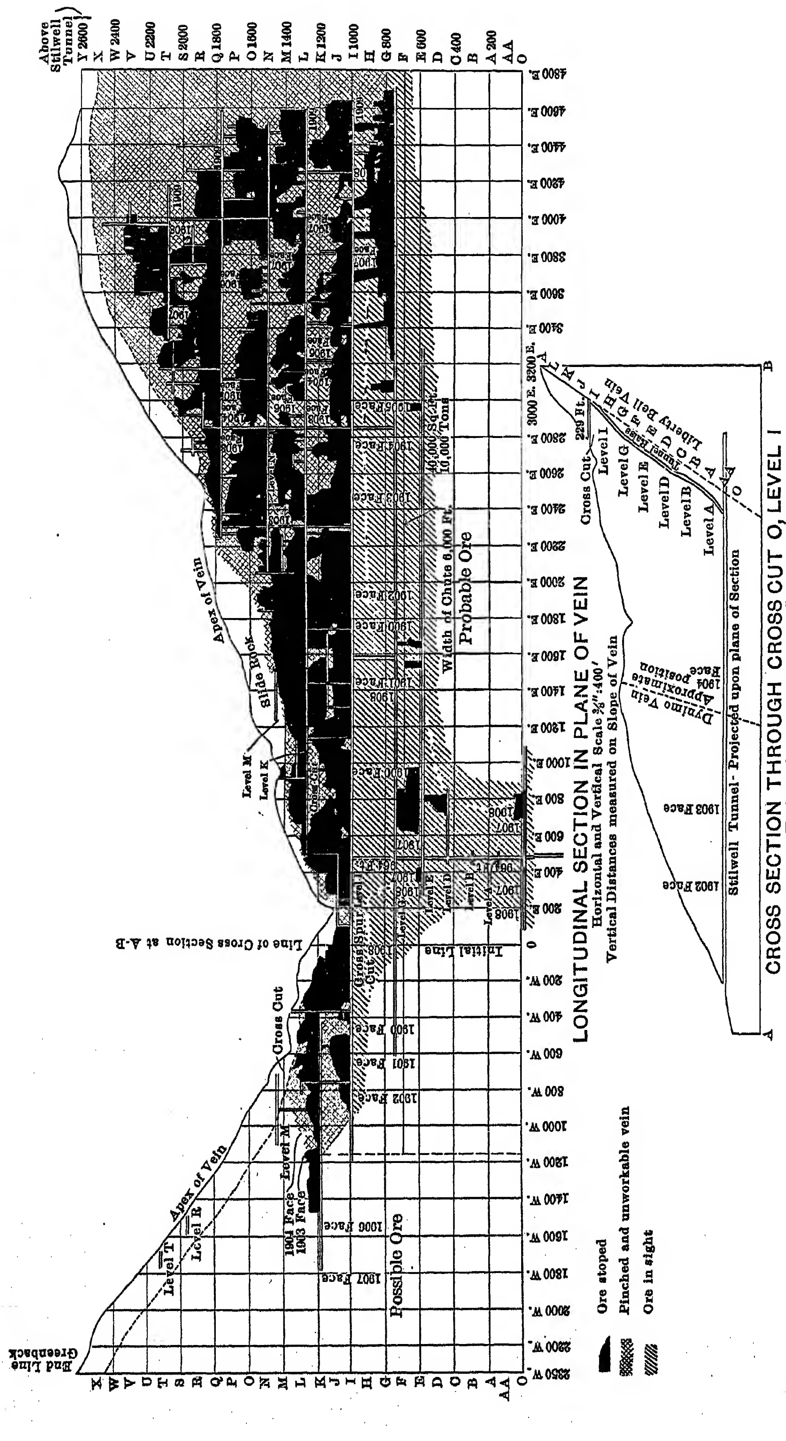

FIG. 1.—LIBERTY BELL GOLD-MINE. LONGITUDINAL SECTION.

The vein is notably uniform through great length and depth. Ore has been stoped for 6,000 ft. along the strike and 2,200 ft. along the dip. Within this area occur local enrichments, varying in size from small, rich pockets to considerable bodies. So far, precise outlines have not been defined. In places, the apparent boundaries have been nearly horizontal, a condition which C. W. Purington reported in 1896 in other mines of the district.[2]

As already remarked, the vein is developed principally in the San Juan series of andesitic tuffs, breccias, and flows. Only a small area is within the higher (Silverton) series of intercalated andesite and rhyolite. A winze on the vein, 225 ft. down from the Stilwell tunnel level, passed the contact between the San Juan formation and the Telluride formation of grits and conglomerates at 125 ft. on the foot-wall and 200 ft. on the hanging-wall. At the lowest point, the vein was narrower, and the gold- and silver-content lower, than above. The vein-filling was largely gouge. These changes signify little in a vein where the distribution of pay-ore is so erratic. This is the first working of the district which has gone down into this lower horizon, directly beneath a largely-productive ore-body in the volcanic rocks above; the vein shows generally less diminution of values with depth than others of the district. The hanging-wall is treacherous; fortunately, it is usually much firmer where the vein is wide.

Mining-Operations.

The longitudinal section of the mine, Fig. 1, shows the system of co-ordinates in use. On the strike, the mine is laid off laterally both ways from the junction of a vertical plane, AB, normal to the strike, with the plane of the vein. On the dip, the succeeding letters represent 100-ft. lifts from the tunnel-level. Chutes and man-ways take their whole number from the block in which they occur, east or west, and their decimal designation from their position in the block. For example, 36.3 chute in G level is 3,630 ft. east of the dividing-plane, and is immediately under 36.3 chute in levels above. This system is patent to all, and is both a help and a safeguard to all men underground.

[2] "A Preliminary Report on the Mining Industry of the Telluride Quadrangle."

The loose, soft vein-filling and the flat dip have controlled the mining-practice. When the mine was first opened, the economy of air-drills in soft, narrow veins was doubted. Certainly, with abundant, skilled hand-miners there could have been little gained by the use of piston-drills in stoping or raising. On the other hand, drifting by hand is slow at best, and the mine has paid, in later years, a heavy penalty for its slow early development. To the SE., the apex is masked by great depths of slide-rock. Therefore, development was from below; and it was always necessary to mine below, before the ground above could be made ready—a misfortune in soft material.

In 1904–5 two new types of power-drills appeared: the Temple electric-air drill; and the Leyner rock-terrier, a small air-hammer machine. The former proved useful and efficient in drifting, and the latter in stoping and raising. Power-mining having been proved a success, an adequate plant was installed; and since that time development has been rapid and systematic, so that to-day the mine is in a well-developed condition.

An undertaking of first importance was the Stilwell tunnel, completed in May, 1905. This adit, driven at an elevation of 10,400 ft., 800 ft. below the lowest (I) level of the upper mine, cut the vein at 2,600 ft. The ore was of paying grade, and uniform in character with that above, giving promise of long life for the mine and warranting plans for a commensurate equipment.

A raise, to become the main artery of the mine, was driven 1,010 ft. to the upper workings (the work consuming one year); and thereafter stations were cut, and lateral development was begun. The plan adopted for the lower mine was a retreating system, mining inward from the lateral extremes and from the top down. To this end, the top, or G, level was driven at high speed, and the ground above is now being mined from the SE. end. This system concentrates a large part of the mine-work on one level at a time. Stope-supervision is particularly effective. The haulage being by electric motor, trammers and station-attendants are reduced to a minimum number. Haulage-ways are maintained in the best possible condition, free from chute-mouths, between the producing

section and the raise, and protected, above and below, by ground in place. Old ground being abandoned as fast as new ground is opened, maintenance-charges should be constant and low.

The methods of mining and timbering are simple. Originally drifts were timbered with square sets, and cribbed two-compartment mill-holes were carried up through stopes filled with waste, shot from the foot-wall. The method was open to these criticisms: (1) the hanging-wall, broken at the drift for the square-sets, was weakened; (2) the ore required shovelers in the stopes; (3) the waste shot from the wall inevitably mixed with the ore and there was a steady loss of ore in the filling; (4) the timber of the cribbing offered serious impediment to the free flow of ore in the chutes; and (5) in addition to the timber for the mill-holes, posts and stulls had to be used to support blocky ground in vein and wall.

Later practice has been to depend almost wholly on stull-timbers, these points being in favor of the change: (1) the unbroken hanging-wall preserves its original strength; (2) the ore is delivered to the chute-mouth with little manual labor; (3) the unbroken foot-wall offers a minimum resistance to the flow of ore; (4) the ore is cleaner and the loss is small; (5) somewhat less timber is required, and uniformly better support is given to bad ground close to the working-faces. Of course, the hanging-wall must cave eventually; and it is necessary—and feasible—to withdraw entirely before this happens.

Stull-timbers were used throughout the length of the tunnel-raise, a practice unusual, but warranted by experience. Ample pillars are in place on both sides. A certain amount of light scaling from the hanging-wall was expected, and has occurred, causing no serious inconvenience. The chief advantage derived lay in the greater free space left open between the walls, affording flexibility in adjusting the track to the changes in dip. All stulls are on 4-ft. centers on the dip, the two central lines being 10-in. by 10-in., and the outer lines, next the pillars, 8-in. by 8-in. square timbers, all painted with carbolineum before being put into the mine. Head-boards are used. The only cross-bracing, except in very wide places, is by track-ties.

The breaking of ore is done almost wholly by Murphy air-hammer stoping-machines (old pattern). In places hand-augers are used to advantage. Holes are drilled at a high pitch, and are shot in series from a free vertical face. Miners work on a partial floor near the back of the stope. Commonly the ore falls through the working-floor upon sloping floors, which deflect it to chutes at from 25- to 35-ft. intervals. As the stope advances, new sloping floors are placed; the vertical lining is built up; and the old sloping floors are ready for re-use.

For this common type of stope, stulls from 8 in. upward, costing 10 cents per foot, are placed 5 ft. apart, in floors 7 ft. apart. The working-floor is made largely of 6-in. round timber at 6 cents per foot, and sloping floors and chute-lining are largely of 10.5 ft. round or split lagging, at 13 cents per piece.

From this typical stope, practice varies, with increasing strength of hanging-wall and higher angle of dip, until half the stulls and both floors and chutes may be omitted, the men standing on broken ore to mine. The final cleaning-out of one of these stopes involves considerable scraping from the foot-wall.

The Murphy drill used has proved particularly fitted for this uneven ground, largely by reason of its small feed-piston, only $1\frac{3}{8}$ in. in diameter. The total pressure exerted on this small area is hardly more than sufficient to hold the steel against the ground, and, if a "fitcher" threatens, control is easy. Almost all the other patterns of air-feed drills have been tried, but without exception the large diameter of their feed-pistons (commonly $1\frac{15}{16}$ in.) drives the drill into hopeless fitchers. The valveless hammer-drills seem better suited to work with the small feed-pistons, by reason of the definite air-cushioning of the hammer, when the chuck is not fully on the steel. (Since the writing of these notes the Ingersoll-Rand Co. drill (MC 22) has demonstrated its fitness. It is valveless, and has small feed-piston area.)

Development-Work.

Drifting is done by contract. For much of it Temple machines have been used; but these have given way in large measure to Sergeant 3.25-in. drills, which are operated on swing-shifts, morning and afternoon, between the two main shifts.

This organization has permitted the full use of compressor and power through 24 hr. (That the Temple-Ingersoll drills gave place to air-drills does not imply their failure. The air being available, and the other machines simpler, they were used. The No. 5 Temple could out-drill, and their power-consumption was not one-quarter that of, the 3.25-in. Sergeants. The drill itself is wonderful in its simplicity and strength. With intelligent and painstaking supervision, the electrical end makes little trouble. This machine has a large field.)

Until recent years, it was the practice to drill any face once a day only, leaving the drift free for shovelers and trackmen on the opposite shift. The presence of the electric motors of the Temple drills made this almost essential. The necessity of crowding some development-work led to the practice of drilling and shooting twice daily, as outlined. The results are satisfactory. It seems as easy to get a good contracting-crew to organize for two shifts, with 200 to 250 ft. of advance, as for half that. The contract-rate is the same. Contractors buy from the company all powder, fuse, caps, and candles, and place stulls and lagging; and their shovelers move the cars to the siding, never more than 500 ft. away. The drift is broken not less than 9 ft. high by 6 ft. wide, and a ditch is carried forward on the hanging-wall side. The regular price for such drifting is $8 per foot, except for widths exceeding 9 ft., for which $11 is paid. Contractors are held to strict accountability for maintaining proper grade and cross-section, and the company agrees in the contract that grade shall be checked by the surveyor on the completion of each 25 ft. of advance. The grade is 0.5 per cent.

The higher speed in drifting serves to concentrate development-work in few headings, simplifying supervision for foreman and surveyor.

In explanation of the practice of compressing and drilling through the 24 hr., it may be said that power is bought on the peak-of-load basis, the highest peak three times recurrent in any month marking the charge for the entire month. On this basis, power is had at $5 per h-p.-month, measured on the high-tension line. Obviously, it is desirable to equalize the load. All stope-miners and timber-men now work on the day-

shift. One hand-miner picks down and loads for two machine-men, thus making the machines effective throughout the shift. The improvement in efficiency is such that the loss in power unused at night is unimportant.

Ore-Movement Underground.

Chute-troubles, more particularly from the ore hanging on the foot-wall, were encountered early. With the advance of mining-work, we came to use a main chute 700 ft. long, which, during the driest months of winter, was choked so as to require reopening every three days in order to permit the movement of ore. Moreover, during the rest of the year, in spite of every reasonable precaution, enough water found its way into this passage to semi-liquefy the soft ore. If the ore was allowed to accumulate, the chute broke, frequently burying trains below, and, in one case (fortunately without loss of life), an entire crew of trammers. After transportation to the surface, this water-logged ore broke tramway-bins and flooded the station; on the way to the mill, it overflowed tramway-buckets; and at the mill, it burst the battery-bins and flooded the batteries. The ore, as it came from the stope, was soft, but reasonably dry; and it was in the main gathering-chutes that this trouble became acute.

The remedy for this condition was found in lowering the ore through the chutes in skips. The initial installation was, of course, an emergency-job, the skip being installed in a three-compartment chute and man-way. Fortunately, this chute was unusually well built, and, while space was limited, and a lateral angle made some trouble, the experiment was successful. The arrangement is very simple. A discarded $\frac{7}{8}$-in. tramway-traction rope leads from the 4-ton skip over a 48-in. Hallidie grip-sheave, and down to the counterweight of cast-iron blocks on a truck. Pockets are not used; the ore being dumped direct from the cars. At the discharge-point, the door opens automatically, and the skip is returned to an upper level by the counterweight.. At the level the tender closes the door by hand, hooking himself about his waist to a chain before he steps into the skip-way. Two brake-bands are used on the head-sheave.

The second installation was in a 300-ft. ore-chute, without room for a counterweight. The empty skip is returned by

motor and the door is opened and closed automatically. The hoisting-cable runs to a spreader above the skip. Over this spreader runs a second piece of cable, the ends going to the two lower corners of the skip-door. The skip stops in hooked rails, and the load forces the door open. After the discharge, the power first closes the door and then raises the skip. This installation also has been a success.

In both these cases, when changes in the dip of the vein require it, the cable is deflected on chilled-iron plates; the consequent wear on the cable being less costly than the maintenance of idler-sheaves in places of difficult access. The skips discharge into pockets between vein-walls, carefully guarded from water. Subsequent transfers, in both cases, are made by trains, drawn by horses or mules.

The success of the first of these gravity-skips led to the reconsideration of a plan to connect the new Stilwell tunnel with the upper mine by a vertical raise, in which cars were to be lowered by cage. This idea was abandoned, and a skip-way on the vein was carefully designed, to serve as the main outlet and inlet during the entire life of the mine. Fig. 2 shows the character of the arrangement.

This being essentially a gravity-plane, power is necessary only when an empty skip is to be lowered or an unusually large load is to be hoisted. By reason of the purchase of power on the "peak-of-load" basis, a motor of only 25 h-p. is provided, which operates the skip at 100 ft. per minute, while on gravity the speed is from 500 to 600 ft. The filler, or small pocket, just under the level, holds one skip-load, 6 tons. The toggle-locked door is opened with a single movement of the lever by the cager on the level. At 100 ft. above the tunnel-level, the skip drops through a switch and falls into a vertical position, dumping automatically; the door being self-locking thereafter.

Transportation for men and timber is furnished by a man-cage coupled above the skip—the two carriers making one six-wheeled unit, and the power being applied at the knuckle between the two elements. The reason for this construction may not be obvious. As originally planned, on the assumption of uniform dip, the carriers were separate units, and power was to be applied at the upper end of the man-cage. But sharp changes in dip were encountered; and, as the man-cage

is usually empty and the skip loaded, the light cage would be lifted heavily against the guides by the pull of the cable. This condition led to the construction adopted, by which the power is applied direct to the heavy load and the light man-carrier is independent of the lifting tendency of the rope. The results have been satisfactory.

The carriers look, and are, light; but they have proved adequate. In operating these gravity-planes, it has been evident

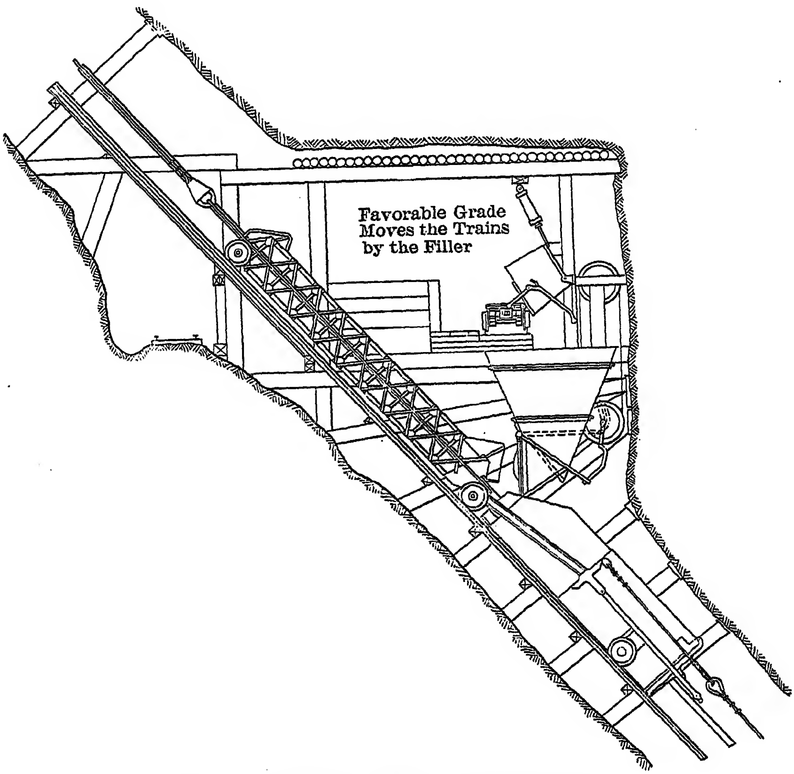

Fig. 2.—Ore-Loading Methods at a Level.

that success lies in having the weight of the equipment constitute the smallest reasonable part of the total weight of ore and equipment. To this end, three novel features were introduced: (1) the skeleton man-cage, holding 28 men and weighing 2,700 lb. (a rope-ladder, dropped into the skip, holds 15 men in addition); (2) the rope-bail, a 50-ft. section of $1\frac{1}{8}$-in. plow-steel cable, spliced endless and passing around curved

cross-heads at the knuckle, and at the attachment to the main cable; and (3) the tubular skip, built of $\frac{1}{4}$-in. steel plates and having a standard $\frac{3}{8}$-in. boiler-head for a door. Of all shapes of skips, the tube has the advantage of greatest capacity and strength per unit of weight. The door is remarkably strong. The skip complete, with liner-plates in its lower third, weighs 3,900 lb., and its capacity is 150 cu. ft. This equipment, with $1\frac{1}{8}$-in. plow-steel cable, full-loaded, has, at the steepest part of the raise, a safety-factor of 5 (old ratings).

A $1\frac{1}{8}$-in. tail-rope passes from the ore-skip around a tension-sheave in the 50-ft. sump and rises to the counterweight of 15,000 lb., thus effecting complete rope-balance at all positions of the carriers.

The endless-rope mechanism for braking and driving is placed with its drums in the upper extension of the plane, the rope not being deflected to or from it. The upper drum, geared to the motor-clutch, carries two slip-rings, and two half-laps of the rope. A lower transfer-drum carries a single ring and one half-lap. Each of two independent hand-operated brakes acts on both drums, providing ample safety. The slip between rope and rings, and between rings and drums, is slight and gradual, and does not prevent the geared indicator from showing the approximate position of the skip. The exact position is shown, however, by red-lead marks on the cable. Idler-sheaves, 48 in. in diameter, with hard-iron wearing-rings babbitted in, support the cable at the changes of dip. Signaling is done by electric bell. Two No. 6 hard-iron wires, 3 in. apart, can be bridged by the cager at any point. The cager is called by telephone. It is not necessary for him to accompany the ore; and he is left free to dump cars into the fillers at the stations.

Haulage on the levels presents little that is new. In the old mine, horses and mules pull the trains on all levels save one, in which a home-made locomotive handles a train of small cars. On level G, the highest of the new mine, a Westinghouse single-motor 3.5-ton D. C. locomotive operates 1.5-ton cars in 3 trains of 12 each. These cars are side-dumping, are carried low, and are hinged far over towards the discharge-side, which makes accidental dumping quite impossible, and hand-dumping impracticable. At the tunnel-raise, the one point where dis-

charge is desired, an air-actuated piston, hooking into a ring fixed on the side of the car, gives almost instantaneous discharge. The door is toggle-locked.

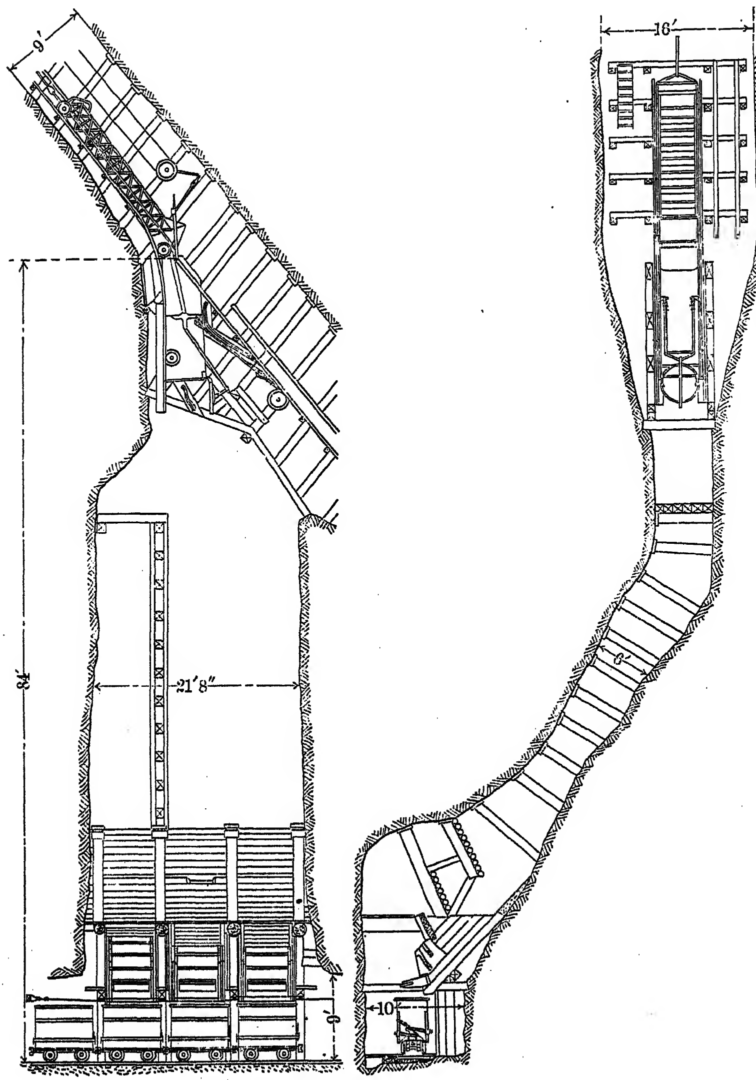

Fig. 3.—Ore-Pocket Arrangement.

For drift-installations, the bonded track-return is not used. Two No. 0 conductors of hard-drawn trolley-wire are placed, 6 in. apart vertically, on the foot-wall side of the drift, passing

under the chute-mouths. Between these runs a three-wheel carriage, one wheel under the upper wire and two on the lower. The upper wheel is insulated from the others, and a spring in tension between the two lower ones holds the upper one in position. Flexible cable connects to the locomotive. The two conductors are in plain sight and give high efficiency, avoiding the constant losses on a bonded track.

Opposite the loading-chutes on this level, the ditch is bridged with plank, and sheet-iron plates are placed between the rails. The bridge prevents the blocking of the ditch with spilled ore, and, with the plates, greatly facilitates the maintenance of a clean track. Petersen automatic track-switches, and a switch for the two-wire trolley-line, greatly assist train-movement.

The ore is dumped in four-car lots into the filler, and thence to the skip, by which it is conveyed to the 200-ton ore-pocket (Fig. 3) in the foot-wall above the level of the Stilwell tunnel. This adit, a tangent, extends into the foot-wall, underneath the pocket. The cross-section, Fig. 3, shows the arrangement. The cars used here hold 73 cu. ft., or 3.2 tons, each; have double-gable bottoms, wheels and axles of railroad type; and toggle-operated doors, and are run in trains of 8, with the locomotive on the outside end. No switching is done in loading or discharging.

The locomotive, like that employed in level G, of which it is a duplicate, has given almost perfect records. Both carry 25-h-p. motors, and operate at 6 miles per hour with full load. The wheels slip at 16 horse-power.

Electric Mine-Plant.

This mine has derived decisive benefit from the availability of "custom" electric power. The demand for considerable amounts of power underground encountered the difficulty of long-distance low-potential transmission, and led to the introduction of high-tension (10,000-volt) circuits, with inside transformers, situated near the places of use, and effecting a reduction to the customary 440-volt distributing-circuits.

The first installation, made in 1906 at the east end of the mine, used an A. S. & W. triple-conductor, rubber-insulated and lead-covered cable. This was placed in a conduit of half-

weight 2-in. iron pipe, known under the trade-name of the "Patent Loricated Conduit." The second installation feeds the main transformer-station at the inner end of the Stilwell tunnel. The cable enters Level I above, and runs down the tunnel-raise. By reason of the great weight of lead, a thick protective covering of tape was used as a substitute. The conductors are No. 14 stranded copper wires. To prevent rupture of the cable from its own weight in the raise, three No. 10 steel wires are laid within the cable, interspaced with the three conductors. The cable is laid in a conduit like that of the earlier installation. Three years' service seems to have proved the fitness of the lighter cable.

The transformers used in the mine are of the common oil-cooled type. That serious danger may attend their use, was evidenced in August, 1909, when lightning entered the old station, breaking down the transformers and firing the oil. The station was burned, and three men subsequently lost their lives in the smoke. This station has gone out of service; the new main station is effectively walled-off; and a man is continuously on duty, to close, in case of danger, all apertures connecting with the rest of the mine.

The secondary alternating-current distribution is effected with ordinary single-conductor double rubber-covered wires. In raises, the wires of any one circuit are placed together in a conduit of the kind used for the primary circuits; and in drifts they are carried open on the stulls. This current is used for compressors, hoists, fans, Temple drills, motor-generator sets, saw, and lights.

Direct current at 250 volts is generated at two points, but principally in the tunnel-station, whence it is carried inward a mile for drift-service, and outward 3,000 ft., for tunnel-service.

An all-important feature underground is the Stromberg-Carlson telephone. Eleven instruments are used. These are warmed by incandescent lights immediately beneath them; and the lead-covered wires are carried in conduits, except in the tunnel, where the line is open. This expensive construction seems essential to the excellent results secured.

The ventilation, ordinarily automatic, is adequate, except in long drift-ends and raises. For these places, small Buffalo or

Sturtevant fans, driven by small motors, are located at the extreme limits of good natural ventilation. The best results are secured when the fan is about midway of the line of ventilating-pipe, drawing through one end and blowing through the other. Next in importance to the supply of air is its control. In adit-ventilation, the draft is commonly disagreeable and may be dangerous—a condition which has been remedied here by the use of a two-door air-lock in the tunnel. The doors are operated by the motorman and one is always closed.

Costs.

The following detailed tabulations of mine-costs show various changes attending the transition from predominant hand-mining in 1906 to machine-mining with mechanical ore-movement.

Supplies.

	1906.	1907	1908.	1909.	1910.	January, 1911.
Product of ore, tons,	92,221	102,429	116,133	126,336	134,321	148,776
	Cents.	Cents.	Cents.	Cents.	Cents.	Cents.
Powder, per ton,	8.76	10.07	10.68	8.79	9.68	9.09
Fuse and caps,	2.20	2.14	2.53	2.38	2.63	2.48
Candles,	3.87	3.79	3.35	2.83	2.63	1.81
Steel and tools,	1.84	2.29	1.78	1.54	1.40	0.38
Blacksmithing,	0 30	0.31	0.39	0.40	0.32	0.30
Timber,	15.12	18.22	19.59	19.26	17.36	14 50
Nails and spikes,	1.34	1.18	1.44	1.17	1.17	0.98
Heating,	3.16	2.70	2.51	3.10	3.16	3.40
Lighting,	0.17	0.30	0.23	0.31	0.51	0.64
Ventilating,	0.28	0.11	0.12	0.19	0.21	
Cars,	1.77	1.82	0.99	1.13	3.28	2.40
Hoists,	0.84	0.60	3.68	2.49	1.28	2.11
Skips,		3.41		4.63	5.66	3.57
Drills,	1.95	4.23	1.30	1.13	1.26	0.56
Air-lines and compressors,	0.57	2.00	2.45	1.54	1.84	0.74
Lubricants,	......	0.70	0.72	0.71	0.90	0.64
Track-supplies,	0.48	0.20	0.38	0.57	0.54	0.08
Electric plant,	1.42	1.17	0.80	2.66	5.84	0.28
Electric power,	4.05	5.41	4.70	6.96	7.20	9.11
Ore-bins,	0.15	0.61	0.11	0.64	0.80	
Phones and signals,	0.15	0.36	0.13	0.16	0.38	0.15
Locomotives,	......	......	......	......	1.45	0.04
Miscellaneous,	0.81	0.45	0.37	0.38	0.38	0.17
Total,	49.	62.	58.	63.	69.	53.

Labor.

	1906. Cents	1907. Cents.	1908 Cents.	1909. Cents	1910 Cents	January, 1911. Cents
Foreman,	2.78	2.58	2.47	2.28	2.14	1.82
Bosses,	7.59	9.10	9.00	6.90	5.16	5.03
Clerk,	0.23	0.40	0.48	0.52	0.75	0.90
Stoping,	70.63	54.05	51.11	34.17	32.14	26.00
Hauling,	39.40	46.57	47.04	47.50	42.77	18.00
Tunnel-raise,	......	......	......	6.40	6.78	6.85
Blacksmithing,	4.01	4.09	5.00	3.84	2.36	2.03
Timbering,	43.03	48.09	44.89	45.40	41.44	28.72
Track-laying and ditching,	2.78	2.70	2.34	3.88	4.30	0.62
Heating,	2.44	2.50	1.70	2.25	2.38	1.49
Phones and signals,	0.04	0.22	0.29	0.37	0.40	0.18
Air-lines and compressors,	1.66	2.04	3.44	3.32	3.81	1.16
Lighting,	......	......	0.10	0.10	0.10	0.05
Ventilating,	0.11	0.20	0.10	0.75	0.30	
Ore-bins,	0.33	0.90	0.37	1.19	0.85	0.83
Electric plant,	0.86	2.03	2.19	2.81	2.69	1.56
Miscellaneous,	0.74	0.61	0.36	0.62	2.13	0.47
Total operating-labor,	177.	176.	171.	162.	151.	96.
Total operating-supplies,	49.	62.	58.	63.	69.	53.
General assays and surveys,	3.	6.	5.	4.	4.	2.
All operating,	229.	244.	234.	229.	224.	151.
Labor,	27.	43.	56.	36.	23.	
Supplies,	15.	14.	17.	12.	7.	
Development-work,	42.	57.	74.	49.	30.	17.
All mine-cost,	271.	301.	308.	278.	254.	168.
Depreciation,	5.	6.	6.	7.	8.	8.
All cost,	276.	307.	314.	285.	262.	176.

The foregoing figures as to labor and supplies are independent of development-work, and hence are comparable from year to year. The expense of development has not been itemized.

Through the period covered by these statements, much of the ore was mined from increasingly-remote workings, a condition which reached its maximum in 1909. In spite of this condition, and of the higher wages paid to machine-miners, and the introduction of the employment of mine-mechanics, the cost of labor per ton of ore shows a progressive decrease. The effect of the hammer-drills on stoping-costs is striking. The figures as to hauling do not show what was really accomplished. In 1906, the maximum handling of ore embraced

two car-movements. In 1909, some ore was moved four times by car and three times by skip; half of the ore had three transfers by car and two by skip; and the nearest ore had two movements by car and one by skip—which will be the general requirement hereafter. The remote upper workings are now cleared. In future, the workings will be compact; and the total cost of mining, including necessary development, should not exceed $1.75 per ton. This figure, it is true, has been attained in but a single month; but in that month 17 per cent. of the ore was from the old mine. Moreover, recent improvements in organization should effect considerable savings.

Boarding-House.

An essential feature of a San Juan mine is the house for the men, Fig. 4. These structures have gradually improved until those built in recent years are highly creditable. The new house of the Liberty Bell at Stilwell tunnel is thoroughly modern. In construction it is a mill-frame of Douglas fir with all joists and girts open, minimizing the opportunity for the accumulation of dirt or vermin, or the origination of fire. The dry-room has 200 open-steel lockers, and shower-baths and toilet-closets. The floor is of concrete. The company operates the dining-room and a commissary-department in the building.

Crusher-House.

The arrangement of this building need not be specially described. The ore, delivered above, is dumped upon steep (52°) 3- by 1-in. bar grizzlies, the coarse sliding down to the crusher, and the fine (together with the crushed product) going to the ore-bin, under the crusher. An unusual area of grizzlies is imperative. Even with the area in use, screening is difficult in the wet season.

The crushers are duplicate Denver Engineering Works 11- by 18-in. sectional machines, the heaviest piece weighing 3,000 lb. Much is to be said in favor of this type for trail-transportation. The drive is from 40-h-p. motor and the crushers are brought to speed with heavy Hill clutches.

The bin-frame is of 8 by 16 in., instead of the common square timbers. They offer greater strength per unit-area of cross-section, and offer less hindrance to the free flow of ore, a valuable

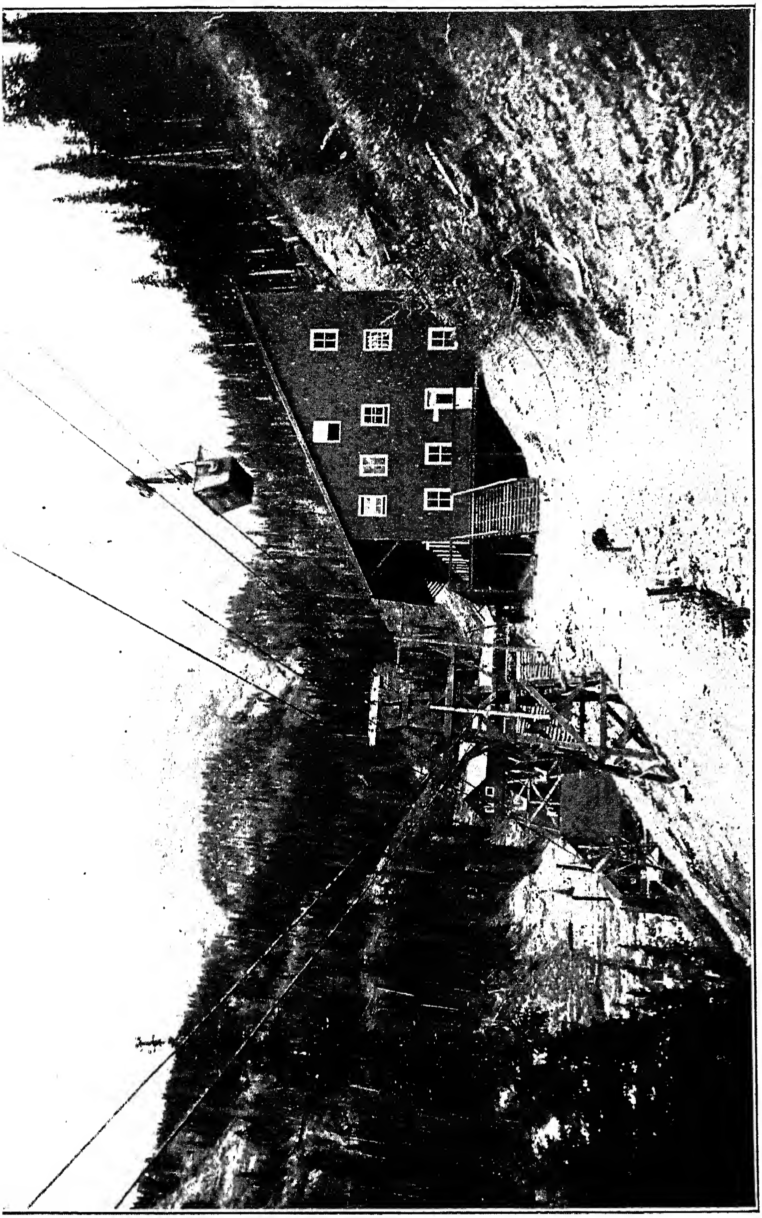

FIG. 4.—LIBERTY BELL BOARDING-HOUSE AND TRAMWAY.

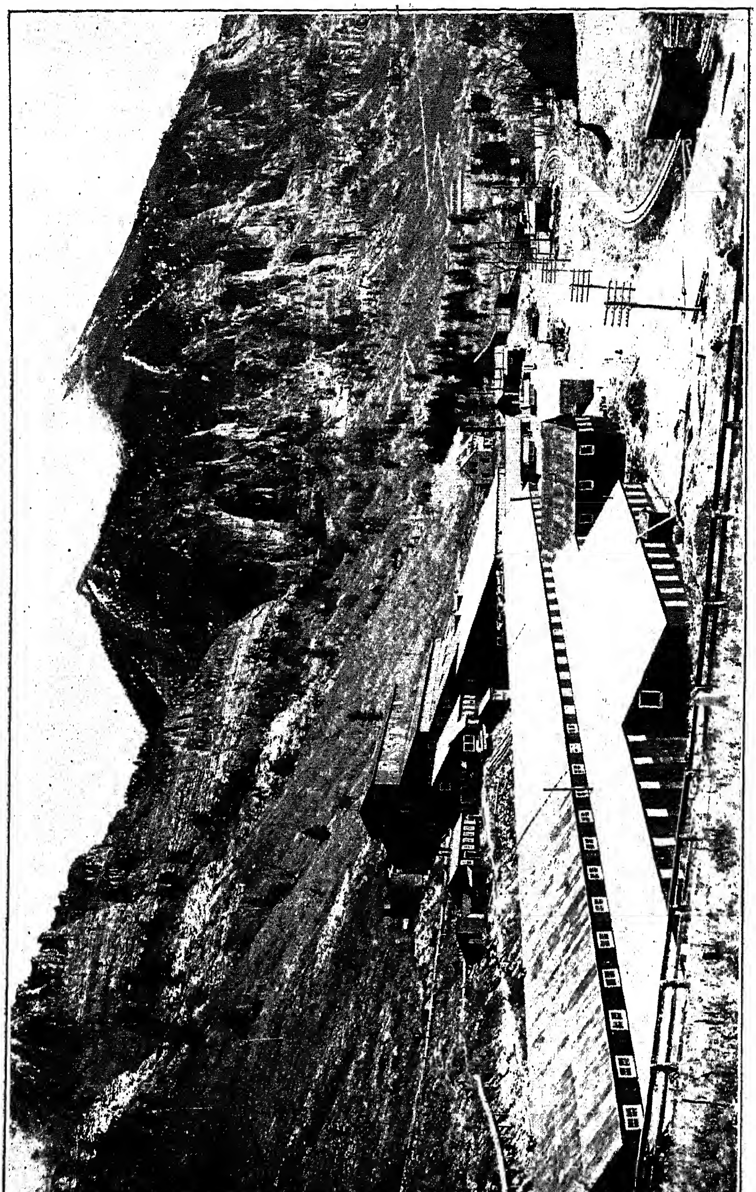

FIG. 7.—LIBERTY BELL MILL.

consideration here. Girts are held in with pressed-steel hangers of the Lane type. These materially cheapen and improve the construction. Below the ore-bin is the tramway-terminal, and below that is the steam heating-plant for both ore-bins and boarding-house. The coal-supply is stocked from the tramway-buckets in a bin below the tramway-floor.

The Aerial Tramway.

This structure, originally a nondescript, has been developed to high efficiency. It was at first 2 miles long, with an angle-station; but on the opening of Stilwell tunnel, the upper half mile was cut off and the angle-station became the upper terminal.

The greatest obstacle to successful operation has been the necessity of crossing a divide, some 400 ft. higher than the present upper terminal, and 1,800 ft. higher than the mill. I imagine that the first tramway of this class was built on lines ideally simple, running down an even slope. Under such conditions certain types of equipment were developed, some of which are maintained to-day. The load carried was the weight of ore, and the weight of the portion of traction-cable carried by each hanger was insignificant. The typical triangular wrought-iron bucket-hanger is adequate for such conditions, but not for a heavy-duty line, crossing a divide. In that case the load of ore (700 lb.), though considerable, is a small matter compared with the weight of the traction-rope, as each bucket raises it, in passing the divide. To meet this harder condition, the cast-steel hanger, Fig. 5, was devised, with such results that shop-work on rolling-stock has practically ceased, while before one hanger per day to the shop for reforming was the average. A good feature of the bucket is the Schuler friction-grip, a successful local invention. The use of a canvas (20-oz.) liner in the pans of the carriers has been found profitable, as insuring complete discharge, and, therefore, full capacity for the carrier, without the necessity of ruinous pounding to clear the pans. A steam-heated detention-room, giving each unit about three minutes' time before dumping, has been found useful in winter, to thaw the carriers that have hung loaded between shifts.

One or two innovations in station-construction have been introduced. At the loading-station, buckets upward-bound, with freight for mine or boarding-house, are shunted to a stub-tramway 200 ft. long, which rises 60 ft. to the mine-portal—the

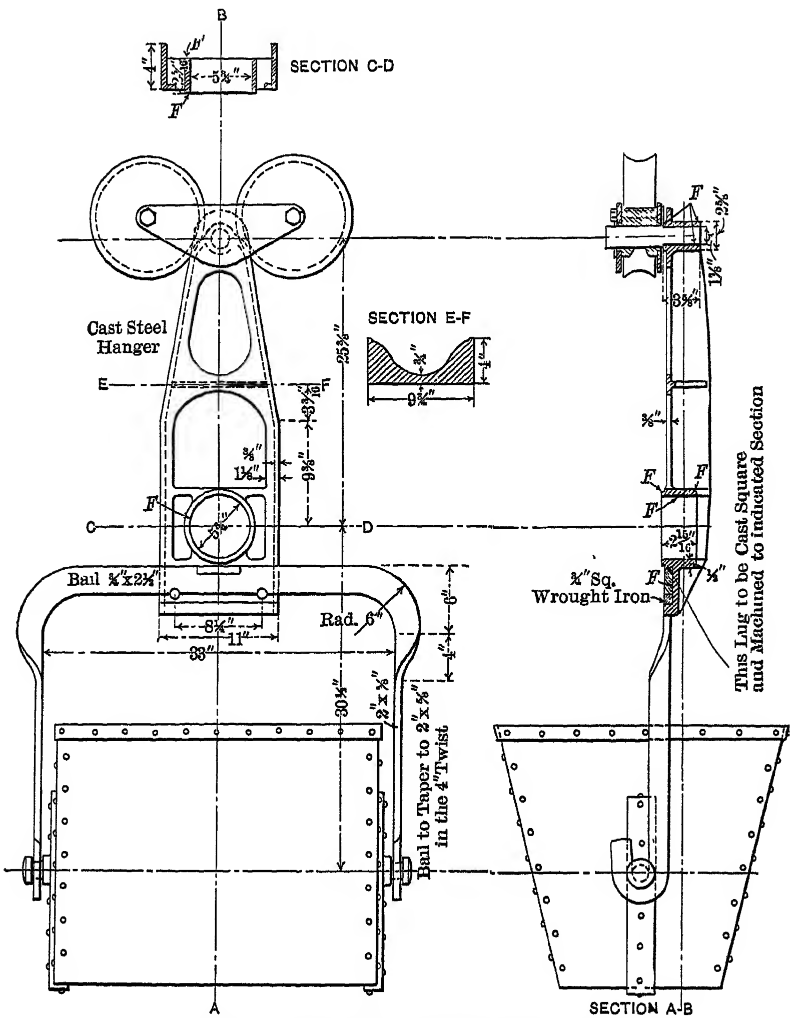

FIG. 5.—CAST-STEEL TRAMWAY-HANGER.

level of the boarding-house and of general storage. The empty bucket returns to the main station, to be loaded with ore. This stub is a complete tramway in itself, driven by gearing from the main line.

At the lower terminal, a two-way station rail-switch of new pattern, Fig. 6, has been introduced.

The tramway passes over one high divide and a minor ridge. At both points, the track-cable is supported by trestle, and relieved of the weight of the carriers. The development of suitable track over these trestles was slow; but the final result is good. We use a 5-in. bulb-angle with cast-iron support. This section, rolled in high-carbon steel, shows practically no wear, and gives a track almost as smooth as the rope itself.

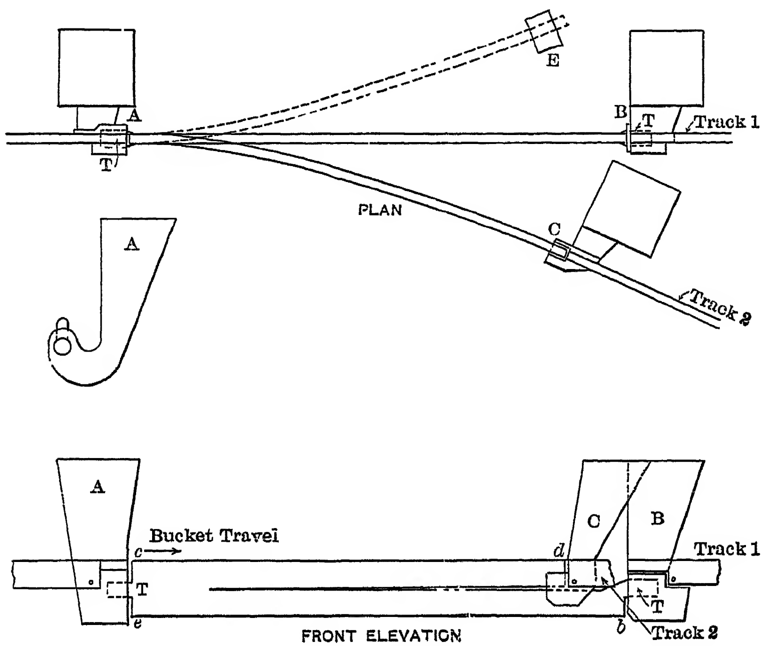

FIG. 6.—TWO-WAY STATION SWITCH.

The approach to the rail, as well as to the saddles on many supports, is over arched sheet-steel shields.

At all high points on the line the usual small-diameter idlers supporting the traction-rope have given way to 30-in. sheaves of bicycle type, with hard-iron bearing-rings babbitted in.

Track-cables of $1\frac{1}{4}$ in. and $1\frac{1}{8}$ in. diameter are used in 0.5-mile sections, carrying 15 and 10 tons respectively as tension-weights. The preservation of maximum tension seems to be prerequisite to the long life of cables.

The traction-rope is a ⅞-in. 6 by 19 Lang lay, Seale patent, special crucible-steel cable. The line is regulated by a Bleichert automatic controller, which governs with great precision where even passably good results were impossible with hand-braking. This machine, built to absorb 50 h-p., is a rotary pump, forcing liquid through a balanced valve, the aperture of which is regulated by a governor in the fly-wheel. The liquid falls back into a closed reservoir, where the heat is removed by water-coils. Nice modifications of speed are secured by adjustments of the valve-stem.

The following figures show costs for five past years and a typical month of the present year.

	1906.	1907.	1908.	1909.	1910	Month, 1911.
Tons,	92,000	102,000	116,000	126,000	134,373	12,500
	Cents.	Cents.	Cents.	Cents.	Cents	Cents
Labor,	23.54	22.05	25.24	20.49	13.56	12.
Supplies,	10.79	9.31	8.84	7.54	4.09	1.4
Total operating,	34.33	31.36	34.08	28.03	17.65	13.4
Depreciation,	2.	2.	2.	2.	2.	2.
All cost,	36.	33.	36.	30.	20.	15.

NOTE.—The sharp reduction in costs after 1909 is in large part due to the simplification and shortening of the line, as a result of which the angle-station became the new upper terminal.

In comparing these costs with the notable 6 and 7 cents costs of the Utah Consolidated tramway, for instance, it is to be observed that this ore weighs 80 lb. per cu. ft. of carrier, and is sticky; while the Utah ore weighs 150 lb. per cu. ft. of carrier, and is free-running. This point is worthy of note by those interested in tramway-costs.

The line was built as a narrow-gauge (6 ft.) line for 15 tons hourly capacity. With many of the original structures still in use, its duty to-day is from 25 to 30 tons per hour. The practical limit in loading the line is the strength of the carrier to lift the traction-rope as it passes the divide. The present standard for the tramway is to load 80 buckets hourly, spacing at 45 sec. and 206 ft. on a line running 275 ft. per min. The bucket complete weighs 400 lb. and the load 700 pounds.

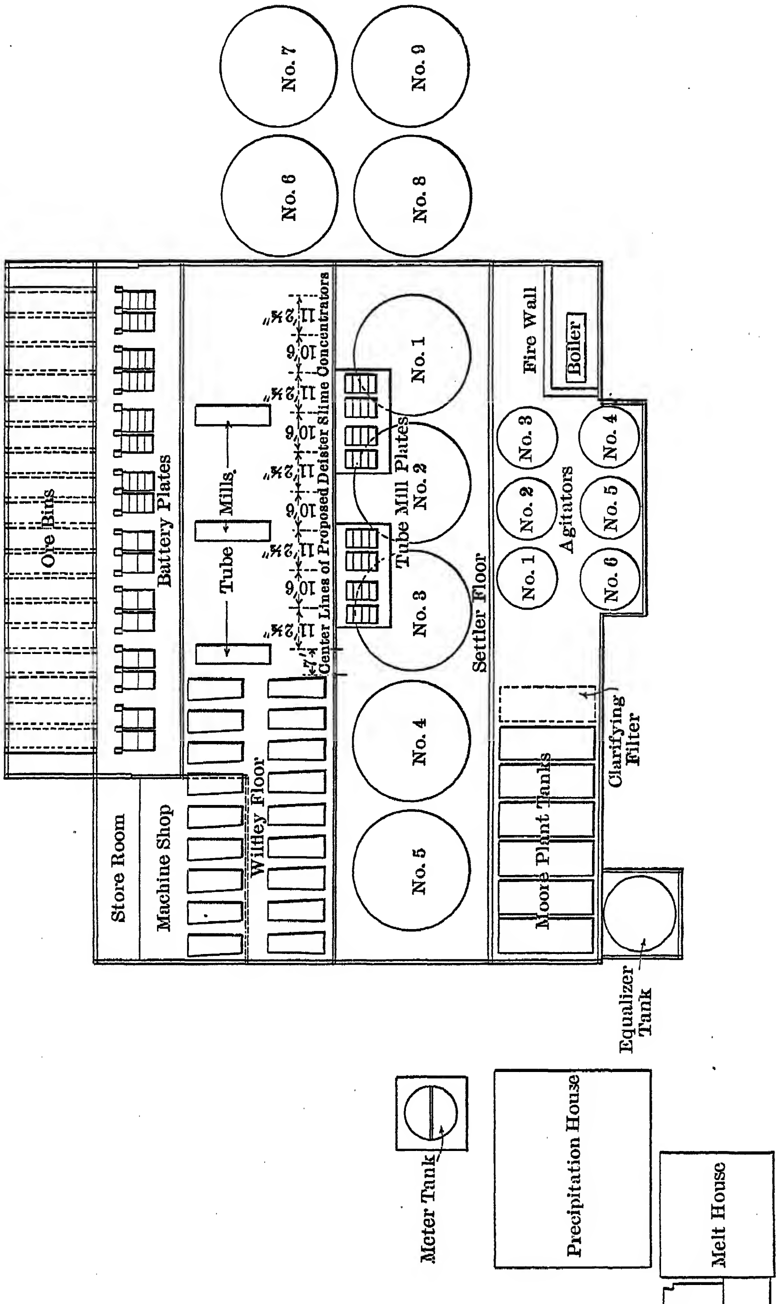

FIG. 8.—LIBERTY BELL MILL. GROUND-PLAN.

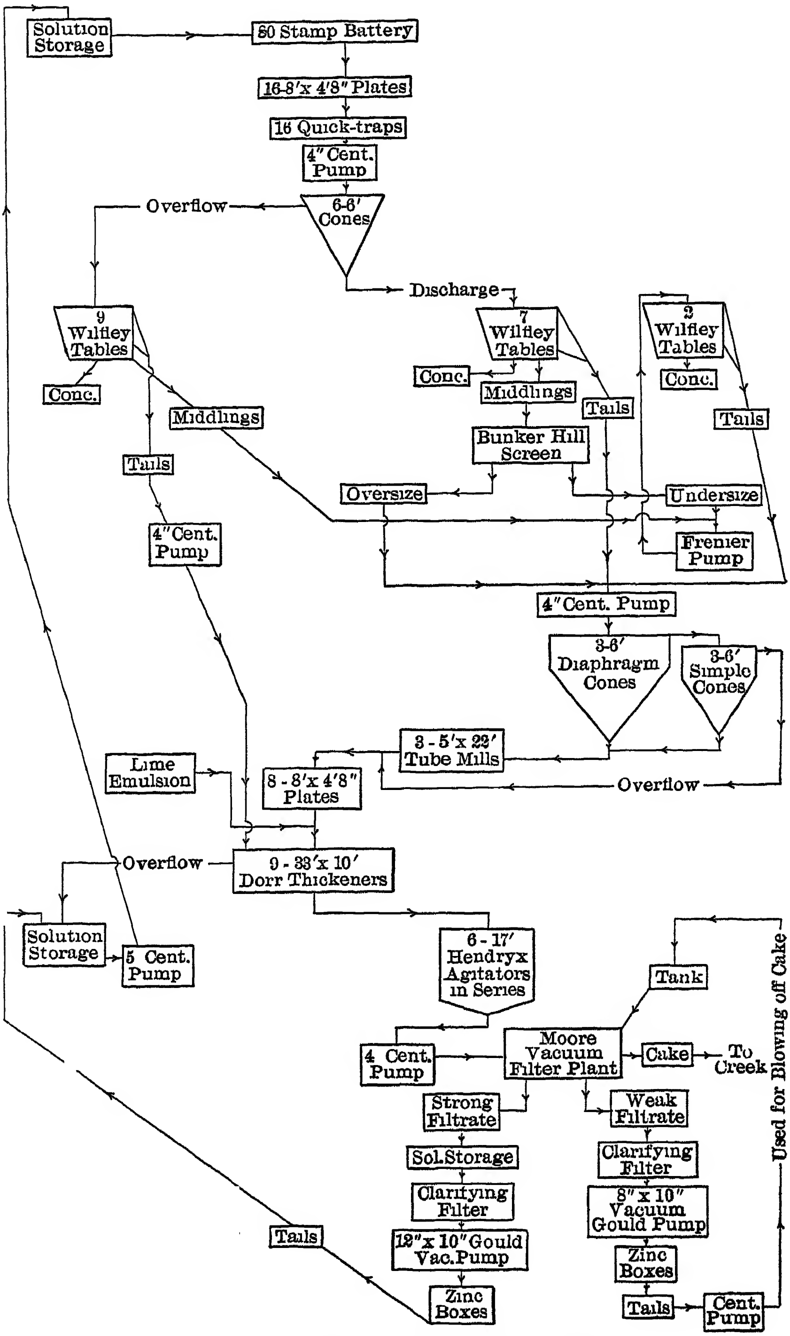

FIG. 9.—FLOW-SHEET, LIBERTY BELL MILL.

The Liberty Bell Mill.

The metallurgical practice at this mill has been so fully described that I need not dwell on well-known features. A general view of the mill and surroundings is given in Fig. 7. The ground-plan, Fig. 8, and the flow-sheet, Fig. 9, show the relations of the following units of equipment:

1. Eighty 850-lb. stamps, with suspended Challenge feeders.
2. Sixteen 8- by 4-ft. copper amalgamating-tables, with three 1-in. drops.
3. Four Richards vortex (hindered settling) three-spigot classifiers.
4. Eighteen Wilfley tables and 10 Deister No. 3 tables.
5. Three 5- by 22-ft. tube-mills of Abbé pattern, the feed being thickened by Dorr classifiers or diaphragm-cones.
6. Eight amalgamating-tables, of the size given above.
7. Nine 33- by 11-ft. Dorr continuous settlers.
8. Six agitators of the Hendrix type, 17 by 11 ft., above the 45° cone.
9. One 20- by 15-ft. equalizer-vat.
10. A Moore filter-plant of seven vats, each 9 by 27 ft. in area, and 8.5 ft. deep to the coning.
11. A zinc-shaving precipitation-plant, containing 1,200 cu. ft. of zinc.

Stamp-Battery.

This battery was originally built on wooden blocks, with the usual framing and a front horizontal drive from clutch-pulleys on the line-shaft. The wooden foundations have given way to concrete, and the framing is simpler. The post goes to the concrete, only a piece of 6-ply Gandy (or similar) belting intervening. The results have been perfect.

Ten stamps have the heavy Allis-Chalmers anvil-block; the others, the lighter "sub-bases" of the Denver Engineering Works. There is no apparent difference in results; and the lighter construction is cheaper. Globe stem-guides have been reasonably satisfactory through many years, but are now giving way to the simpler and stronger Pacific guides. Shoes, boss-heads, and tappets are of chrome-steel. Cams are of the Allis-Chalmers Blanton pattern. The Blanton fastener is used for the bull-wheel. Dies are of cast-iron, from the local foundry, containing a large percentage of steel from scrap.

The horizontal battery-drive through clutch-pulleys was unsatisfactory, and solid pulleys were substituted, it being cheaper and easier to cut an occasional belt, in case of desiring to stop a ten-stamp section for considerable repairs, than to maintain the clutches. The feeders are operated through the feeder-

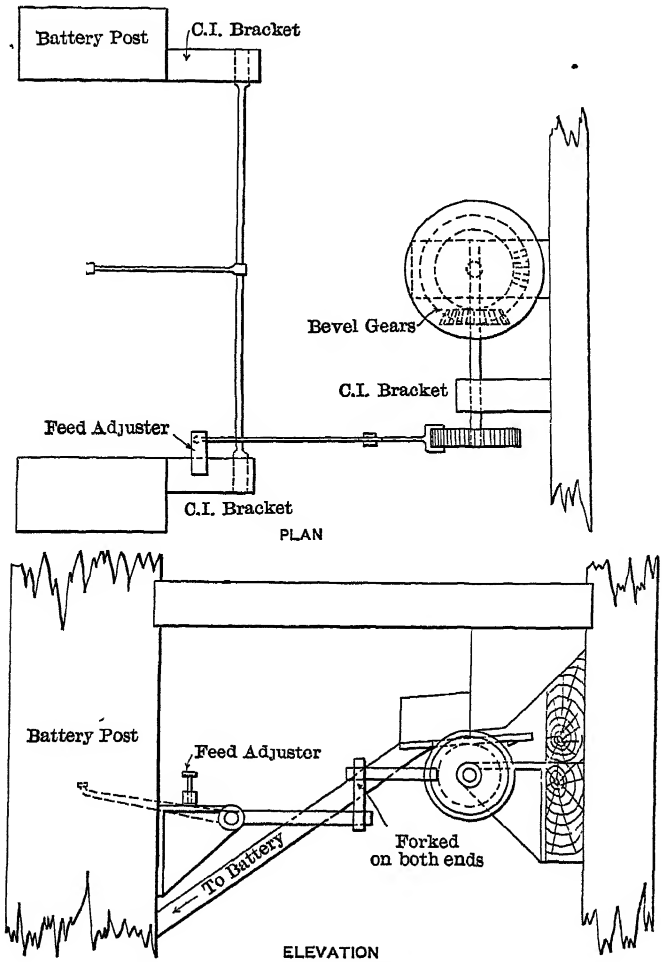

FIG. 10.—BATTERY-FEEDER MECHANISM.

wheel, Fig. 10—an unusually good device, patented by the mill-foreman in 1900.

Battery-screens in recent months have been of two patterns, namely: 14 by 14 sq. mesh No. 22 wire, aperture, 0.043 in.; and 16 by 3 mesh Ton-cap, 0.039-in. aperture. The latter

yields a finer product and a larger tonnage than the former, the Ton-cap having heavier wire and the same aperture blinded. It seems that the wire must be light enough to spring readily under the impact of the splash. A screen-analysis of battery-pulp shows: on 40-mesh, 24.4; on 60-mesh, 10.9; on 80-mesh, 5.9; on 100-mesh, 6.3; on 200-mesh, 9.6; and through 200-mesh, 42.9 per cent. As determined by centrifugal test, 38 per cent. of the battery-pulp is flocculent.

The power charged to the stamps is 160 horse-power.

Amalgamation and Concentration.

The ore is stamped in cyanide solution. The recovery by amalgamation is materially smaller than in previous years of water-amalgamation; the process is more expensive in both labor and material, and requires more skill; and the consumption of copper is considerable. Muntz-metal, which has proved a satisfactory substitute for copper elsewhere, has not been satisfactory here, by reason of the hard, glassy surface which it assumes. The plates are kept rather wet; and any drip of quicksilver is caught in a trap. From a month's run the results of amalgamation were:

From battery-plates, 80 per cent. of all amalgam, yielding 29 per cent. of bullion; fineness, 0.408 Au and 0.551 Ag.

From tube-mill plates, 20 per cent. of all amalgam, yielding 23 per cent. of bullion (0.153 Au and 0.825 Ag).

The grade of the first plates is 2.25 in., and that of the second plates 1⅝ in., per foot.

The concentration-scheme is only now assuming definite form. Nine Wilfleys take the underflow from the Richards classifiers; two take the middlings (after the coarse has been removed on a Bunker Hill screen); and seven take the overflow, after thickening in six 6-ft. cones. The ten Deisters are to take the reground sand from the tube-mills. Any oversize tailings from the latter tables are returned to the tube-mills.

This arrangement seems to represent a reasonable economic limit. Further expansion of plant would yield relatively slow returns on the investment. The great impediment to perfect work is the argillaceous slime, which, coagulated by the alkaline solution, is exceedingly buoyant, and sustains coarse material, both sulphide and sand, until dilution has been carried to

extreme limits. The above scheme represents the lesser of two evils. For three years, concentration was applied to the tailings, after filtration and dilution with water. An extensive area of canvas, with Wilfleys and vanners, gave poor returns; and it was evident that the sulphides, probably concentrating to some extent in the tube-mills, had been ground so fine as to be irrecoverable. Moreover, what was caught was so high in grade as to suggest re-precipitation of silver on the pyrite. The extraction of gold was comparatively good.

The continued practice of amalgamation at this mill has occasioned some adverse comment. It is recognized that the omission of this step would materially cheapen and simplify the milling. Sixty stamps, at most, would crush the full tonnage through the coarser screens that could be used. But the gold occurs irregularly in the ore, and hence is likely to be coarse; and this gold would make an unwelcome element in the concentrates, which have not as yet been made amenable to local treatment. A streak of gold on the tables would be a constant source of possible loss; and the product would be spotted, and difficult to sample for sale.

Were it feasible to concentrate successfully after regrinding, the battery-plates might well be done away with, and the coarse gold allowed to go into the tube-mills, to be ground and taken into solution; but in this case, it seems to have been demonstrated that concentration after regrinding is not good practice.

The power charged to concentrating is 30 horse-power.

Regrinding.

The tube-mills are of the Abbé type, tire-mounted and with spiral feed. The tire-mountings, designed by the company's engineers, are amply rigid, and also give complete protection against possible end-travel of the tires off the rollers. The tires, both on the mill and on the supporting rollers, are of forged steel and promise indefinite life. The mills are driven by 50-h-p. motors, belted to counter-shafts connected through heavy friction-clutches with the tube-mill shafts. The mills start readily with the use of the clutch, showing a maximum starting-peak of 78 h-p.; the running-load varies from 45 to 48 h-p. The lining, of 4-in. silex set in cement mortar, with the narrow edge to the wear, lasts for 12 months, contin-

uous service. The grinders are 4-in. imported flints, costing $33 per long ton delivered. The ends are lined with local cast-iron; and the discharge is through a grating, which will probably give way to the Neal cone-discharge.

Typical Screen-Test.

Screen-Mesh.	Feed. Per Cent.	Discharge. Per Cent.
On 40	47.1	0.7
On 80	29.4	11.4
On 100	8.2	9.2
On 200	6.4	22.3
Through 200	8.9	55.4

About one-half of the total tonnage crushed is reground. The efficiency of regrinding varies with the degree of the previous removal of the slime, the presence of which gives buoyancy to the pulp within, and makes the grinding poor. From this stand-point, the above screen-test is not satisfactory. The practice is to force a diaphragm-cone and to care for the overflow of sand in a simple cone in series. The discharge from the simple cone, which carries an excessive proportion of slime, is mixed with the discharge from the diaphragm-cone, to dilute it to 48 per cent. of moisture. The result is the loss of much of the benefit derived from the diaphragm-cone. It will undoubtedly prove better to combine the overflow from the three diaphragm-cones in a single simple cone, and confine the hampering effect of the fine material to one mill; or a Dorr classifier, at this point, would be still better. The feed to the other mills being diluted with clear solution, the grinding should then be good.

The work of this diaphragm-cone in preparing feed for a tube is remarkable, as the following screen-test shows. The completeness of the elimination of small sizes suggests the benefits of hindered-settling. The cone in this case is 6 ft. deep with 60° sides. The diaphragm is 13 in. above the point, with 1.5 in. annular space. The diameter of discharge is 1.25 inches.

Screen-Mesh.	Cone-Feed. Per Cent.	Discharge. Per Cent.	Overflow. Per Cent.
On 40	27.22	58.95	0.19
On 80	24.82	30.66	8.97
On 100	7.55	5.15	7.44
On 200	8.89	3.42	17.
Through 200	31.52	1.82	66.40
Moisture	....	30.8	

No detailed statement of results from the Dorr classifiers is here given. The two machines here were the first erected after Mr. Dorr's original installation at Terry, and being built to fit the available space, they conformed to his pattern neither in area nor in bottom-slope, and were overloaded in operation. Yet the results have been good through four years' service, though not as good as could have been secured from the larger machines.

The pebbles are fed through the day by the shift-boss, being shoveled into the spiral feed; 135 lb. is the daily charge. The mills run smoothly; and the cost of maintenance is very small. The charge for power is 43 h-p. per unit.

Dorr Continuous Settlers.

The last great improvement in the mill was the change to continuous from intermittent settling, in thickening the pulp for agitation. It is not possible to determine the exact results of this change. Decided gains were shown by the experimental unit, and great benefit followed the complete change; but this was partly due to other changes made at the same time. The principal advantages of the new arrangement are: (1) continuous extraction is secured during the period in which, under the previous system, the solutions were inactive or re-precipitating; (2) a given volume of settler-space has 25 or 50 per cent. increased capacity, when thus operated continuously; (3) the continuous extraction in the settlers has given additional value to the plant for settling, as supplementing any deficiency of agitator-capacity; and (4) labor has been reduced by one man on each of three 8-hr. shifts.

This plant, originally of five vats, settling the pulp from the ratio of 5 : 1 to 2.5 : 1, has been increased to nine vats, settling from the ratio of 9 : 1 to 2 : 1. The increase of solution has come with the interpolation, after the battery, of the concentrating-plant, with its great volume of solution for washing and classifying.

The four settlers recently installed have been placed outdoors, with individual conical roofs and underneath shaft-drive in conduit, with great saving in cost as compared with the usual mill structure. The power consumed is 0.2 h-p. per unit.

Agitation.

After extensive experiment, the agitator-capacity was proportioned to the use of a low-potential electric current, to hasten extraction; but the plan of using such a current failed, and, without that feature, the space provided proved inadequate. This has been remedied, in large measure, by the additions to the settler-plant already mentioned. The connecting of all agitators in series for continuous operation was a natural sequence of the adoption of continuous settling. The results appear to be better; but data for exact comparison with the previous charge-agitation are lacking. Some saving in labor and maintenance is evident.

The agitators operate steadily with little attention and very low cost of repairs; but the unit-size is too small for a large-tonnage plant and the power-consumption (from 6 to 7 h-p.) is out of proportion, as compared with Pachuca tank-practice, or arm-agitation, as practiced at El Oro. No benefit was found in spreading the pulp over distributors from the top of the central well; and it is now allowed to plunge from the collar of the well. The power-charge is 50 horse-power.

The Moore Filter-Plant.

The equalizer, Fig. 11, an integral part of the filter-plant, is a simple type of slow-speed agitator, equally efficient for all depths of pulp in the vat, and economical of power. It has been lately patented and put upon the market as the Gordon agitator.

The filter-baskets of 66 leaves, each presenting two 8- by 6-ft. free-filtering surfaces, are carried on two 10-in. longitudinal I-beams, supported by 6-in. transverse beams which extend to the vat-walls. The leaves are of No. 6 (20-oz.) canvas, reinforced on both sides at the bottom of the vertical stitching with a 3-in. strip of the same canvas. The vertical seams are on 2-in. centers, and the wooden strips between are $\frac{5}{8}$ by 0.5 in. The frame is of 0.75-in. iron pipe on ends and bottom, and the top is of strap- and angle-iron. No cocoa-matting or other filler is used.

The leaves are made at the mill, the sewing (No. 4 linen thread, in 0.25-in. stitches) being by power-machine (Singer 7-7), and cost complete $12 each; new canvas alone in place costing

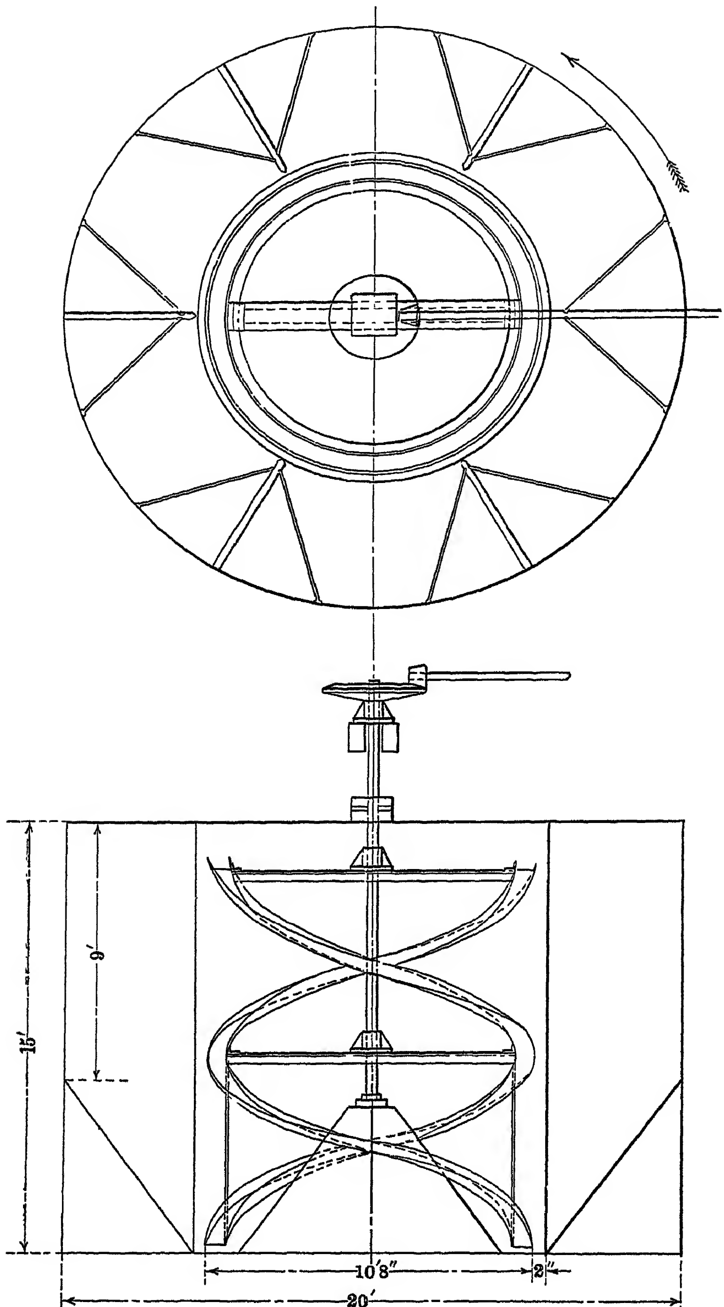

FIG. 11.—EQUALIZER-TANK.

$8. The life of a filter is 18 months. All canvas requires every three months treatment with HCl acid, for which purpose, the basket is immersed in a wash-water vat containing 1.25 per cent. HCl (18° B.) at 140° F.; the liquor being circulated with a wet-vacuum pump. All the canvas in use can be treated in 30 hr. The cost of acid is 0.6 cent per ton of ore.

The baskets are lifted by hydraulic cranes with 20-in. by 9-ft. cylinders, and a pressure of 250 lb. Connection from crane to water-main is made by specially reinforced "quick-as-wink" coupling on a short length of metallic hose. The raised basket is held by a safety-catch on the crane; and the transfer is effected by a 10-h-p., three-phase, constant-speed motor, with a three-armed trolley above. Transmission from motor to crane is done through a Dodge multiple-disk friction-clutch on the motor-shaft. The service is severe; but the clutch does well. The vacuum-connection from the basket, through a 3-in. hose to a pipe turning in a stuffing-box, is maintained throughout the transfer.

The greatest single improvement in this plant was the change from the common wet-vacuum pumps to a combined dry-and-wet vacuum-system, in which all entrained air is taken out at the upper end by a dry-vacuum pump (an 8.5- by 10-in. vertical duplex air-compressor), and all solution at the lower end by centrifugal pumps in a sump 23 ft. below the top of the filters. Some leaks in the canvas will occur; but sand enough to destroy a positive wet-vacuum pump in a few hours is harmless to the centrifugal. The diagram, Fig. 12, shows the arrangement. Between the dry column at one end and one solution-column at the other, runs the main for current strong solution. Parallel, and leading from the same dry column, runs the weak-solution main, but to a different solution-column and pump. Any basket may be connected with either vacuum-main or the blow-off water-main, without disconnecting the hose. The vacuum is held at from 19 to 20 in. (near the maximum attainable at the altitude of the mill), and never fails. The excellent valve designed to secure this result is shown in Fig. 13. It looks like a globe-valve; and its merit lies in seating an iron cone in a hard-rubber ring of square section. It is impossible for sand to lodge on the ring so as to interfere with good closing.

Filtration is accomplished in two groups of three vats each, the central one for loading and the other two for displacement in water. Basket No. 1 loads in the center vat and is moved to the wash-water vat at the right. Immediately thereafter, basket No. 2 moves from the wash-water vat at the left to the loading-vat. The cycle consumes for loading, 50; for transferring and drying, 5; for displacing and discharging, from 45 to 55; and for transferring, 5 min. Each load is a 0.75-in.

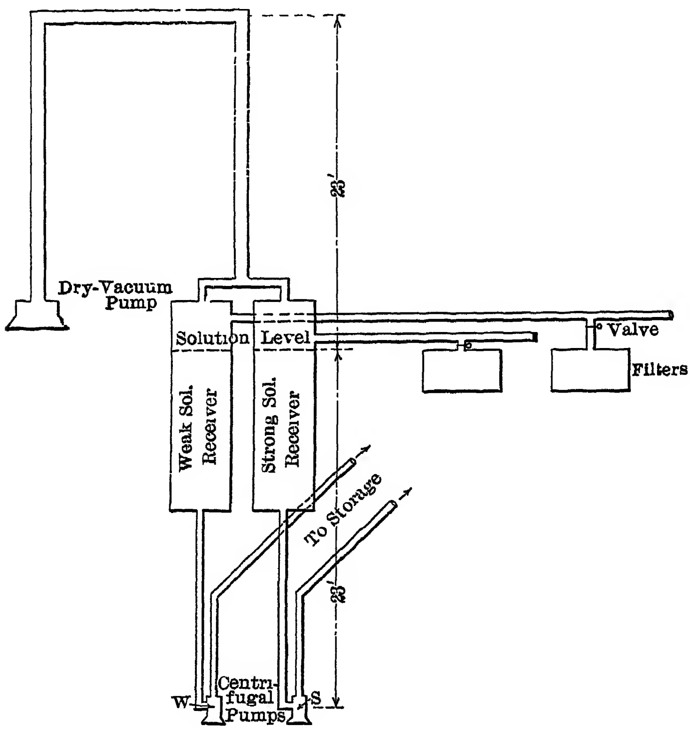

FIG. 12.—DIAGRAM OF VACUUM-CONNECTIONS FOR MOORE FILTER-PLANT.

cake, weighing 2.75 lb. dry per sq. ft., or 9 tons per basket-load. This gives a capacity of 108 tons per basket-day and 432 tons for the plant. Vertical uniformity in loading is secured by three air-lifts, which elevate pulp from the bottom of the vats and discharge it over the top.

The practice of displacing at once in water, without an intermediate wash of barren weak solution, can be approved ordinarily only on the assumption of good displacement and

low-strength solutions. Displacement is here efficient, but the solution (from 1.6 to 1.75 lb. of KCN, at this point) is stronger than was planned when the plant was designed. A factor in this special problem is the 8 per cent. of moisture brought to the mill in the ore.

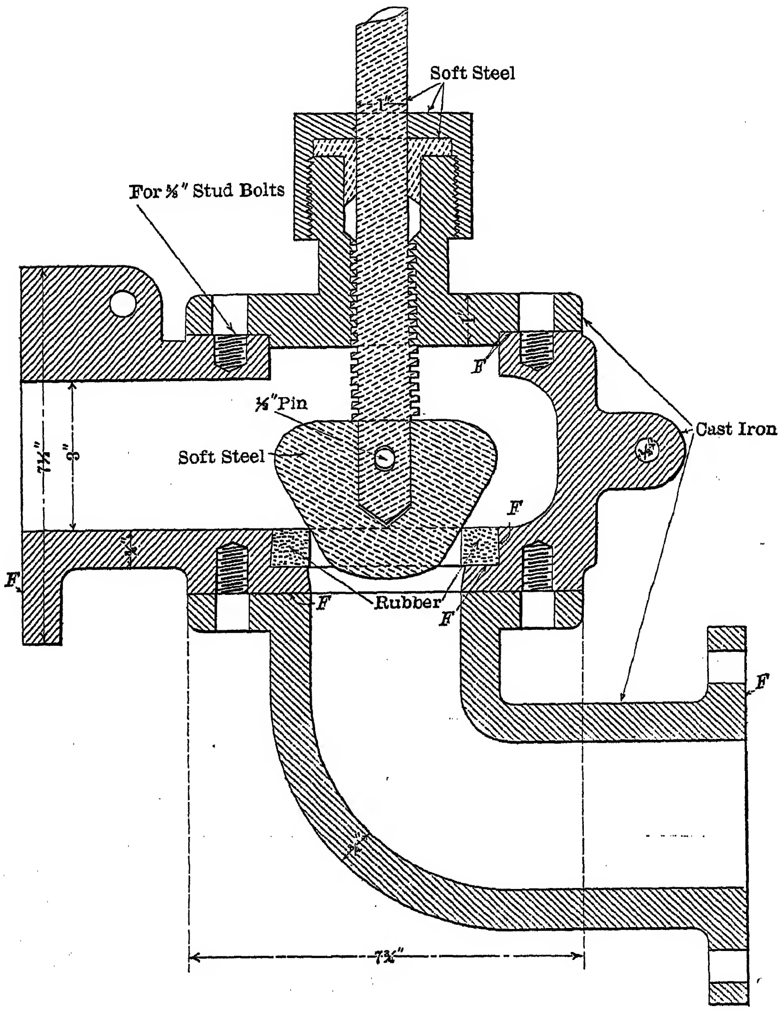

FIG. 13.—VACUUM-VALVE.

Average results in washing are shown by the curve, Fig. 14. The cake, partly dried, contains 33 per cent. of moisture—not taking into account the solution in the pipes and channels, which is difficult to determine, but must approximate 1 ton.

The cake and passages thus contain a total of 5.5 tons of solution, with 8.8 lb. of cyanide and $6.16 per ton in gold and silver. The rate of displacement is 0.15 ton per minute.

In displacement there are two principal objects: the recovery

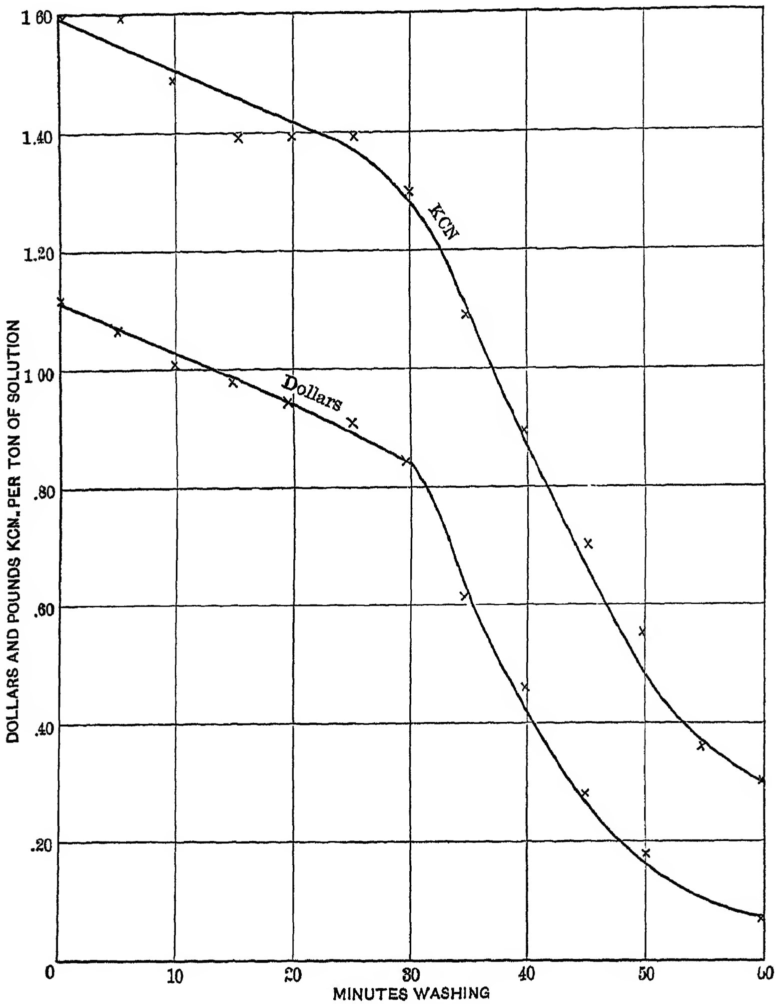

FIG. 14.—FILTER WASHING-CURVE.

of enough strong solution to restore the mill-stock; and the recovery of the dissolved gold and silver in a solution of such strength as to insure precipitation. For the first purpose, 36 min. filtration would be required, if the ore were dry on enter-

ing the mill. The solution drawn in this period contains 7.72 lb. of KCN, so that the efficiency of displacement is 87 per cent., measured in cyanide; and since the metal-value is $5.08 per ton, the efficiency is 82 per cent. measured in metals. It seems fair to accept 84 or 85 per cent. as the efficiency.

The mechanical loss of cyanide by dilution is that which cannot be restored to the mill-stock. The fact that 35 tons of water is brought to the mill with the ore makes it impossible to secure the maximum theoretical efficiency. From each basket-load the solution tonnage recoverable becomes $(5.5 - 0.8) = 4.7$ tons, containing 6.84 lb. of KCN. The combined mechanical loss is therefore $\frac{8.8 - 6.8}{9} = 0.22$ lb. of KCN, or $0.047 in cyanide per ton of dry ore.

As to the second object above named: the recovery of gold and silver in 55 min. of washing is $5.93, the apparent loss in dissolved metals being $0.025 per ton of dry ore. This seems to be a maximum figure; washing for 70 min. showing an almost complete removal. Regular sampling of solution in washed cakes is not convenient; but, so far as it has been done, it shows from $0.01 to $0.02 as the value per ton.

The loss in cyanide by dilution being so small, and the recovery of dissolved metals so nearly complete, the only remaining consideration is the low average strength, 0.75 lb. of KCN, of the weak solution. Solution at 0.9 KCN precipitates well. The use of a barren wash would insure this strength in the weak solution. On the other hand, $0.01 in cyanide per ton of ore will restore the few tons of weak solution to sufficient strength on the infrequent occasions of poor precipitation.

It seems that added costs in depreciation, operation, and maintenance would offset any gains from an intermediate wash.

The weak solution, after precipitation, is used at low pressure, to force the cake from the filter, submerged in wash-water. An advantageous change would probably be to perform this with air, and thus return a more nearly dry basket to the loading-vat.

The removal of tailings is wholly automatic, by reason of the excess of wash-water available. The vat-walls run down to three points, across which discharge single jets of water from

0.25-in. nozzles, carrying the descending mud through from 0.5- to 0.75-in. orifices in the walls opposite the jets.

The operation of four baskets has been described. The fifth is used as a clarifying filter in the seventh vat. All solution, whether decanted or filtered, though apparently clear, requires clarification to insure clean zinc-boxes. In this service, the canvas acquires a remarkably fine, impervious, and tenacious coating. To remove this, the basket is returned to pulp-filtration after from 10 to 14 days. At times, a coat of pulp has been gathered on the canvas before using it to clarify; but this

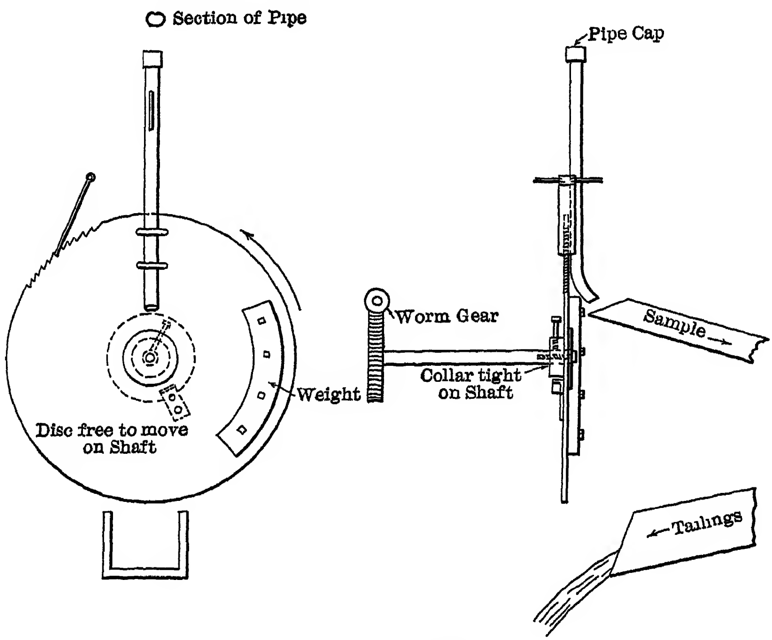

Fig. 15.—Pulp-Sampler.

seems an unnecessary refinement, which reduces capacity. The charge for power is 30 h-p. The pulp-sampler is illustrated in Fig. 15.

Zinc-Precipitation.

This operation shows little that is unusual. The results are usually excellent, with average heads of $1.25 and tailings of from $0.01 to $0.02. The flow of solution is 0.7 ton daily per cu. ft. of zinc, and 2.4 tons per ton of ore milled, 0.3 of the whole mill-solution being precipitated. All solution is metered above the gold-solution vats by a mechanism devised from the

common tilting-box tailings-sampler, Fig. 16. The pans on pipe-guides are always submerged, and steady the movement of the box, after the manner of dash-pots. Since they are placed over the vats, any splash is accounted for in calibrating. Each cycle is registered. The home-made zinc-lathe turns out, per 8-hr. shift, 700 lb. of shavings 0.001 in. thick, which are gath-

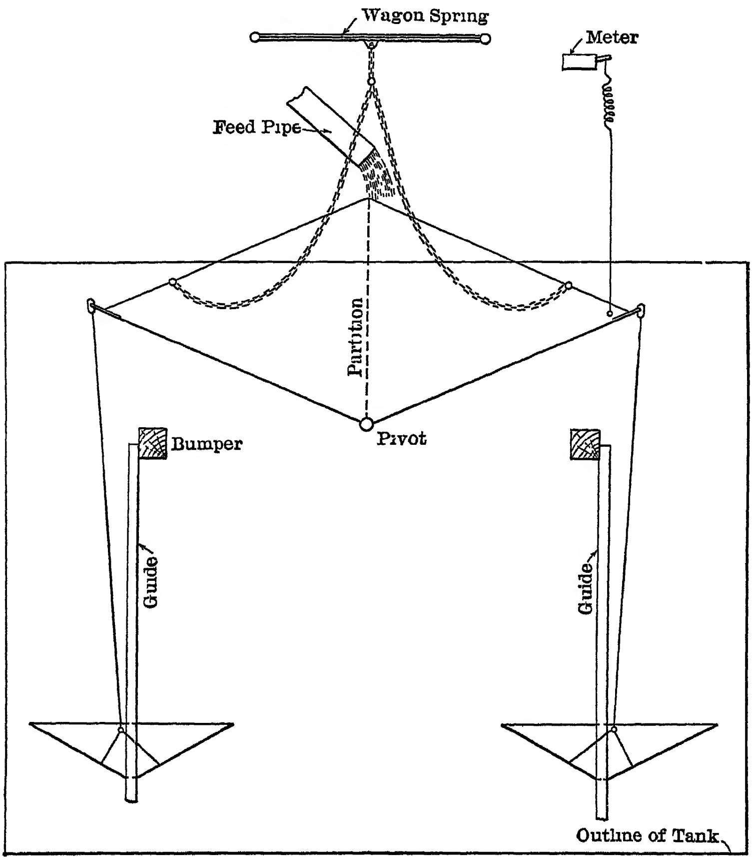

FIG. 16.—SOLUTION-METER.

ered on revolving arms in skeins which fit the boxes. The sludge is collected semi-monthly, treated with sulphuric acid, washed, dried, and melted. Always high-grade, it has recently reached a maximum of 92 per cent. of bullion. The drying-furnace has a cast-iron muffle, and melting is done in No. 150 graphite crucibles, in coke-furnaces. The charge for power in 7 horse-power.

Pumping-Plant.

Pumping required by the multifarious handling of ore and solution is practically all done with an improved Byron-Jackson slime-pump. The pumping-units are in duplicates for pulp; and the plant is described as follows:

Details of Pumps and Service.

No.	Service.	Pump.	Speed Rev Per Min.	Lift. Ft.	Life of Liner. Days.	Life of Runner Days.	Life of Shaft. Days.
1.	To classifier, ...	4-in. B. J.	875	20	40	80	40
2.	Middlings returned,	10-in. Frenier		17			
3.	Sand to tube-mills, . .	4-in. B. J.	910	32	21	63	42
4.	Deister feed, ...	4-in. B. J.	750	10	180	Indef.	120
5.	Agitators to equalizer, .	4-in. B. J.	780	31	60	90	22
6.	Wet vacuum to storage,	4-in. B. J.	955	34	Indef.	Indef.	Indef.
6a.	Same for weak sol., . .	2-in. B. J.	.....	34	Indef.	Indef.	Indef.
7.	Storage to precip., .	Piston-pump					
8.	To mill-feed, . . .	5-in. B. J.	.....	75	Indef.	Indef.	Indef.

The liners used are from $\frac{1}{4}$- to $\frac{3}{8}$-in., cast-iron. Manganese-steel is to be tried. As it stands, the record shows a good centrifugal pump. The table, giving the life of the liners as varying with the thickness of the pulp and the proportion of clay to sand, indicates the cushioning-effect of the clay. The tube-mill feed of nearly clean sand is at one extreme, and the very thick agitator-discharge, carrying all the clay, at the other.

Notwithstanding the good service from centrifugal pumps, I have had for some years the opinion that the proper pumping-equipment for this mill would be a low-pressure compressor, with air-lifts for almost all the transfers mentioned, and for the mechanical agitation. The supersedure of motors, belting, and shafting, with their need of skilled supervision, would far outweigh the loss of efficiency in the air-lifts; and the milling-operations would then be extremely simple.

Use of Chemicals.

The mill-sheets show the following average in cyanide and reagents used:

Battery-Head.		Second-Plate Tailings.		Filter-Heads.		Consumption.		
KCN.	P. A.	KCN	P. A.	KCN.	P. A.	KCN.	CaO.	PbO.
1.75	2.32	1.64	1.44	1.50	1.92	1.48	7.5	0.33

The mixed salt, 99 per cent. of KCN, is used, no advantage being evident in trials of the 130-per cent. salt. A recent con-

cession by the makers has led to further trial with the 130-per cent. salt, with better results. Durango lime is used, and the figures are in equivalents of caustic soda. Until September, 1909, lead acetate was used, from 0.25 to 0.30 lb. per ton of ore being added at the agitators. Since that time, litharge (0.33 lb. per ton) has been added at the tube-mill feed. An apparent improvement of 5 per cent. in silver-extraction from the charge is shown by inconclusive tests.

Cost of Operation.

The following figures cover the period from the beginning of operations with the present type of plant. I do not attempt to give more than the total department-costs, including the expenses readily chargeable to the various departments, and leaving other items as a general charge. The "general charge" for power covers the power for pumping between departments.

The first two years were marked by many mechanical difficulties. The benefit of the abandonment of the canvas-plant and the change to continuous settling was felt in the middle of 1909. The increase in freight, treatment, etc., in 1910, is due to the increase in the tonnage of concentrates.

Year	1906.	1907.	1908.	1909.	1910.	January. 1911.
Tons per year:	92,900	102,106	116,353	125,681	133,881	149,760
Tons monthly average.	7,742	8,509	9,696	10,473	11,157	12,480

Labor.

	Cents.	Cents.	Cents.	Cents.	Cents.	Cents.
Superintendent,	1.47	2.03	1.91	1.98	1.85	1.60
Heating,	0.90	1.33	1.34	1.61	1.40	2.48
Electric plant,	1.30	0.76	1.07	1.05	0.76	0.60
Lubricating,	0.54	0.41	0.24	0.32	0.25	0.26
Accidents,	0.54	1.33				
Pumping-plant,	......	......	......	......	1.65	1.83
Watchman,	0.17	0.13	0.85	1.20	1.03	1.00
Examination and tests,	......	......	2.72	0.13	0.30	0.54
Total general labor,	4.93	6.00	8.56	6.27	7.24	8.33
Crushing,	2.48	3.16	2.01	4.50	4.80	5.35
Stamping,	18.04	17.28	16.66	14.31	13.70	10.95
Regrinding,	4.94	1.46	0.76	0.66	1.28	1.22
Settling and agitating,	5.20	4.51	4.70	2.75	2.36	2.03
Filtering,	13.65	7.18	6.18	5.24	4.46	4.00
Concentrating,	12.11	11.57	12.27	7.02	5.40	5.05
Amalgamating,	5.71	5.44	4.63	4.42	4.67	5.30
Precipitating,	6.68	4.43	3.42	2.83	2.05	1.96
Total labor,	73.75	61.03	59.56	48.01	46.04	44.19

Supplies.

	Cents	Cents.	Cents.	Cents.	Cents	Cents.
Pipe-lines,	0.61	1.81	0.52	0.43	1.30	0.25
Bins,	0.89	0.58	0.05	0.37	0.03	
Building,	3.19	2.36	2.72	3.38	3.73	2.79
Electric plant,	2.79	1.44	1.13	0.86	0.48	0.74
Pumping-plant,	1.93	1.53	1.01	2.46	1.97	3.13
Heating-plant,	4 97	2.80	2.25	2.55	2.58	5.10
Tools,	1.00	0.68	0.22	0.36	0.47	1.02
Cyanide,	40.72	34.19	37.10	31.29	33.90	30.93
Alkali,	5.83	5.91	6.43	5 71	6.45	4.62
Lead salts,	1.22	3.95	4.09	2.85	2.70	4.18
Power,	2.15	3.80	2.96	2.71	2.51	2.24
Light,	1.10	0.42	1.67	1.73	1.65	1.48
Oil and waste,	0.65	1.15	1.00	1.07	0.81	0.86
Assays and melts,	4.17	5.64	4.91	4.00	4.16	4.08
Examinations and tests,	......	1.24	1.59	0.26	0.17	0.07
Miscellaneous,	0.23	0.09	0.06	0.04	0.05	0.05
Total general,	71.37	67.79	67.63	60.24	63.	60 77
Crushing,	4.66	4.80	2.90	2.22	1.77	1.96
Stamping,	17.57	16.85	13.00	13.36	17.80	14.25
Regrinding,	14.89	10.58	8.19	7.05	6.58	6.05
Settling and agitating,	4.95	3.00	3.54	5.20	4.34	5.74
Filtering,	13.80	10.98	6.00	5.59	7.06	5.88
Concentrating,	3.69	3.30	2.85	4.59	3.00	1.87
Amalgamating,	4.93	5.53	4.77	3.42	4.04	2.06
Precipitating,	7.75	7.76	6.43	5.68	6.43	5.44
All supplies,	143.72	130.41	115.28	107.35	114.00	104.03
All labor,	73.75	61.03	59.56	48.01	46.	44.19
Total operating-costs,	217.47	191.44	174.84	155.36	160.	148.22
Depreciation,	16.	16.	13.	11.	13.	13.Est.
Freight, treatment, and discounts,	25.	25.	24.	19.	32.	32.Est.
Total metallurgical cost,	258.	232.	212.	185.	205.	193.Est

Mill-Efficiency.

With the gradual decrease in operating-costs, the percentage of extraction has been raised. The actual results are as follows:

Table of Extraction.

Year	1906	1907.	1908	1909	1910.	January 1911
Gold headings, oz. per ton,	0.351	0.297	0.312	0.242	0.311	
Per cent. recovered by amalgamation,	67	62	56	64	57	
Per cent. recovered by concentration,	1	1	2	4	8	
Per cent. recovered by cyanidation,	22	28	33	28	28	
Total,	90	91	91	92 (?)	93	94
Silver headings, oz. per ton,	5.51	4.63	4.98	3.05	3.45	
Per cent. recovered by amalgamation,	10	9	8	7	7	
Per cent. recovered by concentration,	9	16	14	12	19	
Per cent. recovered by cyanidation,	21	23	27	31	34	
Total,	40	48	49	50	60	60
Total value pe1 ton,	$8.83	9.34	9.16	6.78	8.34	
Per cent. recovered by amalgamation,	48	43	41	45	46	
Per cent. recovered by concentiation,	4	7	6	6	11	
Per cent. recovered by cyanidation,	22	26	32	31	29	
Total,	74	76	79	82	86	88

Summary of Costs.

	1906.	1907.	1908.	1909.	1910	January. 1911	
Mine-production,	$2 30	2 46	2.34	2.29	2.25	1.51	
Mine-development,	.42	.57	.74	.49	.30	.17	
Mine-depreciation,	.05	.06	.06	.07	.08	.08	
Mine total,	2.77	3.09	3.14	2.85	2.63	1.76	
Tramway-operation,	.34	.31	.34	.28	.18	.13	
Tramway-depreciation,	.02	.02	.02	.02	.02	.02	
Tramway total,	.36	.33	.36	.30	.20	.15	
Mill-operation,	2.17	1.91	1.75	1.55	1.60	1.48	
Mill-depreciation,	.16	.16	.13	.11	.13	.13	Est.
Mill-product charges,	.25	.25	.24	.19	.32	.32	
Mill total,	2.58	2.32	2.12	1.85	2.05	1.93	

	1906	1907	1908.	1909.	1910.	January. 1911
Salaries and office,	$0.40	39	.28	.25	.25	
Insurance,	.06	06	.06	.06	.06	
Taxes,	.12	07	.10	.07	.08	
Miscellaneous,	.13	.19	.06	.01	.02	
General expense, total,	.71	.71	.47 (?)	.39	.41	.39 Est.
Total cost,	$6.41	6.45	6.09	5.39	5.29	4.23
Charges to construction,	.49	.24	.36	.37	.36	.36 Est.
Total expenditures,	6.90	6.69	6.45	5.76	5.65	4.59
Credit miscellaneous receipts other than ore,	.14	.12	.26	.33	.21	.21
Net expenditures,	$6.76	6.57	6.19	5.43	5.44	4.38

These figures are given as a service to the public, following the practice of Mr. Winslow in making public his annual reports on the mine. It is hoped that others may derive from them some return for the benefits which the author and his associates have derived from published accounts, letters, and free access to plants elsewhere. Moreover, it is hoped that the figures may serve as a warning in some cases and a source of encouragement in others. Certainly, a mine-manager embarking in a new enterprise under similarly hard conditions should be able to get from this record some measure of the obstacles likely to be encountered. On the other hand, the great improvement of the past two years, here recorded, shows what is possible in that direction. Primarily, this result is the culmination of the plans of years towards the retreating-system and the concentration of operations in mining—a consummation delayed principally by the harassing labor-conditions of 1903–1908, and, in part, by the lack of adequate early development.

Since this mine is situated in a part of the West noted for high freight-rates, living-costs, and wage-scales (averaging, in this case, for mine and mill, $3.60 and $3.75, respectively, per 8-hr. shift), it furnishes an interesting comparison with the results secured from the alleged "cheap" labor of Mexico. This comparison is apt, because, in its metallurgical requirements, the Liberty Bell mine is more nearly comparable with El Oro and Guanajuato than with anything north of the Mexican boundary.

In closing, I wish to acknowledge the courtesy of Arthur Winslow, in granting me permission to publish the many details given, and the valuable assistance given me in the acquirement of special information by W. H. Staver, M. L. Anderson, W. E. Tracy, and H. G. McClain, all of Telluride, Colo.

Rapid Estimation of Available Calcium Oxide in Lime Used in the Cyanide Process.

BY LUTHER W. BAHNEY,* STANFORD UNIVERSITY, CAL.

(San Francisco Meeting, October, 1911.)

LIME is the alkali that is almost universally added to the solutions in the cyanide process of gold- and silver-extraction for maintaining the so-called "protective" alkalinity. It is produced by the burning of limestone.

The value of lime for this purpose depends upon the percentage of calcium oxide contained, which is determined by three factors: 1, purity of the limestone used; 2, degree of the burning-temperature and the period of burning; and 3, length of time of storage of burned material, its condition when stored, and whether it has been damp or wet during the storage.

These three factors render uncertain the quality of lime bought in the open market.

In the United States, lime bought from reliable manufacturers, who thoroughly burn a pure limestone and deliver at once to the consumer from the kilns, may be of a fairly-high and uniform composition; but in Mexico and Central America, where it is purchased from many small producers, who often start with a poor grade of limestone and burn it in small crude kilns with as little fuel as possible, the quality of the product is quite variable.

In consideration of the foregoing, it is apparent at once that there is a great need for a rapid technical method for the valuation of the lime to be used in a cyanide-plant.

The determination of calcium by the gravimetric method, with the necessity of determining also the proportion of carbon dioxide, silica, and iron, requires too much time, and is usually out of the question for an isolated plant unequipped with a

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skilled chemist and the necessary apparatus. The calculating of all the calcium so found to calcium oxide, although sometimes done, is manifestly very inaccurate.

Several methods of titration by means of a standard acid have been described, and no doubt give results sufficiently accurate for a technical method, but the objections to these methods are that they involve the preparation of a standard solution of some acid, usually decinormal hydrochloric acid, which cannot be weighed out, but must be standardized with some other standard solution. Solutions of the following acids have been used by different operators for standardization: sulphuric, nitric, hydrochloric, and oxalic. Oxalic acid is perhaps the most favorable for this purpose, because a standard solution can be prepared by weighing the solid acid and dissolving in water. The use of the solution employed to determine the alkalinity of the cyanide solutions has also been suggested.

While the method of standardization with oxalic acid is open to the objection that the hydration of the acid may vary somewhat, yet it yields a solution sufficiently accurate for technical work.

For the purpose of determining the feasibility of using oxalic acid, the crystals were dissolved in distilled water, and a decinormal solution made. A decinormal solution of pure hydrochloric acid with distilled water was also made, and both were standardized with a solution of chemically-pure sodium carbonate.

Pure calcium oxide was prepared by grinding pure white crystals of calcite in an agate mortar and igniting the fine material in a platinum crucible over a strong blast until constant weight resulted.

This oxide, cooled in a desiccator, was ground in an agate mortar to pass 200-mesh, and the percentage of calcium oxide determined gravimetrically; the result was 99.98, as compared with the theoretical 100 per cent.

The calcium oxide so prepared was used as a standard throughout the succeeding tests. Similar weighed portions were titrated with decinormal hydrochloric acid and oxalic acid, using phenolphthalein as an indicator, requiring 44.2 cc. of hydrochloric acid or 44.6 cc. of oxalic acid to complete the reaction.

The solution of oxalic acid used in the subsequent experiments was made by dissolving 14.6068 g. in enough dis-

tilled water to make a liter, this strength being recently suggested for determining the protective alkalinity of cyanide solutions.

The first experiments were made upon small amounts of 140 mg., to which was added 100 cc. of water before titration, the idea being to have just enough lime present to be theoretically soluble in that amount of water.

This quantity is somewhat small to handle conveniently, and the published method[1] of weighing out 14 g., making 1,000 cc. of emulsion, removing 100 cc. and again diluting to 1,000 cc. and removing 100 cc. for titration, did not give results which checked upon low-grade limes; moreover, this latter method is open to the objection of extra manipulation. A larger amount was then tried, introduced directly into a flask in which the determination was to be made.

The weight of lime to be taken was calculated so that each cubic centimeter of oxalic acid solution would represent 1 per cent. of calcium oxide, as given in the formula:

$$\underset{\text{Lime}}{56.09} : \underset{\text{Lime}}{x} :: \underset{\text{Oxalic}}{126.048} : \underset{\text{Oxalic}}{1.46068}, \text{ in which } x = 650.$$

This weight, 650 mg., was used in all the tests, and Table I. shows the results, which are sufficiently satisfactory for a technical method.

The titrations were made in the cold by introducing 650 mg. of the sample into a 300-cc. Erlenmeyer flask containing 50 cc. of distilled water, using phenolphthalein as an indicator.

TABLE I.—*Results of Titration-Tests for Calcium Oxide, Using Oxalic Acid.*

Calcium Carbonate Present. Per Cent.	Calcium Oxide Present. Per Cent.	Calcium Oxide Determined Per Cent.	Calcium Carbonate Present. Per Cent.	Calcium Oxide Present. Per Cent	Calcium Oxide Determined. Per Cent.
95	5	5.2	45	55	54.5
90	10	10.3	40	60	59 9
85	15	15.3	35	65	64.8
80	20	20.3	30	70	69.6
75	25	25.0	25	75	74.5
70	30	30.2	20	80	80.2
65	35	35.0	15	85	84.8
60	40	40.0	10	90	90.0
55	45	45.0	5	95	94.7
50	50	49.8	0	100	100.0

[1] Treadwell and Hall, vol. ii., p 453.

The results given in Table I. indicate that calcium oxide in the presence of calcium carbonate can be determined by this method with a fair degree of accuracy.

Silica, present in most limes, does not interfere. Magnesia, also present in most limes in greater or lesser amount, is very slightly soluble in water,[2] and shows a faint reaction with the indicator; but it is of no value as an alkali in cyanide-work and should not be shown in a determination of the available alkali in lime to be used for that purpose.

Fortunately, the point where the alkalinity due to calcium oxide stops is readily recognized after a little practice, for the color is a vivid pink, while that of magnesium oxide is faint. Moreover, the color in the titration of magnesium oxide disappears with the addition of only 0.1 or 0.2 cc. of oxalic acid solution, and returns very slowly and feebly, while that of lime is rapid and sharp. This is illustrated by the fact that a titration of pure calcium oxide requires only 5 min., while the same amount of magnesium oxide requires 3.5 hours.

In order to test the oxalic acid titration in the presence of magnesia, two samples of limestones containing magnesia were ground to 200-mesh, ignited in a platinum crucible to constant weight, and titrated. The calcium oxide in each sample was determined by the gravimetric method, since there was no silica present, and only a trace of iron. The following results were obtained:

	Amount of CaO by Gravimetric Method. Per Cent.	Amount of CaO by Oxalic Acid Method. Per Cent.
Sample No. 1, . .	57.6	57.6
Sample No. 2, . .	50.4	51.0

These results indicate that the magnesia does not interfere. Its presence can be judged by the behavior of the titration, and the approximate amount can be quite accurately estimated by continuing the titration, if one has the time needed.

Iron oxide in considerable amount is sometimes present in impure limes, and it obscures or masks the color of the indicator, but if the precipitate be allowed to subside the titration may be carried out to within 1 per cent. of the correct result.

[2] Comey's *Dictionary of Solubilities.*

The determination of the amount of carbonate present in an imperfectly burned lime may be carried on as follows: Grind the sample to pass 200-mesh, weigh out 650 mg. and make the titration in the usual manner; call this result No. 1, "Available Calcium Oxide." Ignite 650 mg. of the finely-ground sample in a muffle or over a blast-lamp, and make a second determination; call this result No. 2. Subtract No. 1 from No. 2, divide by 1.78, and the result will be the amount of carbonate present.

Details of the Method.

The sample must be ground to pass through a 200-mesh screen. Into a 300-cc. Erlenmeyer flask place 50 cc. of distilled water; then add the 650 mg. of the finely-ground sample, stopper the flask, and shake vigorously for 10 sec.; add two drops of solution of phenolphthalein, and then run in the standard solution of oxalic acid until the pink color is discharged; then replace the stopper and again shake. When the color returns, if it is due to lime it will be a bright, vivid pink, and the addition of perhaps 0.5 cc. of solution will be necessary to discharge this color, but if the flask is again shaken and the color is a faint, weak pink returning slowly, this is the end-point for the lime, and indicates that the magnesia is asserting itself.

At all times during the addition of the oxalic acid solution the flask should be violently shaken, being careful not to allow any of the solution to splash out, so the calcium oxide will pass into solution. In nearly every instance of titration of a high-grade lime, the pink color remained vivid nearly to the finish, which shows that the calcium oxide is rapidly soluble.

If a complete titration is allowed to stand for from 15 to 30 min. the pink color will return and show as brightly as in the beginning.

The reading of the burette is in percentage of calcium oxide.

The solutions necessary are: Oxalic acid, 14.6068 g. of pure crystals dissolved in enough water to make a liter of solution. Phenolphthalein, 0.5 g. dissolved in 50 cc. of alcohol and 50 cc. of water.

Electrolytic Oxygen in Cyanide Solutions.

BY T. H. ALDRICH, JR., BIRMINGHAM, ALA.

(San Francisco Meeting, October, 1911)

THERE are two conditions generally prevailing upon the earth—those within atmospheric influence, tending towards oxidation, and those away from atmospheric influence, tending towards reduction. Practically all mineral substances from mines of any depth are in a reducing condition.

Since the cyanide process, in order to dissolve silver or gold, requires that the prevailing conditions under which it operates shall be oxidizing, and the materials usually acted upon being of a reducing character, it becomes necessary to supply oxygen to the solution carrying the cyanide. This oxygen is usually supplied through the medium of dissolved air in the solution, or through the medium of various chemical compounds, which upon combining with the solution or the ore give off a part of their oxygen.

Strange as it may seem, practically all mineral substances are partly soluble in water, especially water carrying alkali or cyanide. The greater the surface exposed and the finer the material is ground, the greater will be the rate of dissolving of the reducing-agents from the ore into the cyanide solution. In most cases, if the solution carrying the ore particles is agitated with air, the air will dissolve into the solution faster than will the reducing-agents; but in some cases the reducing-agents will dissolve more rapidly on account of easy solubility or greater surface exposed. It is a dissolving race between the oxygen from the air and the reducing-agents from the ore, and if the reducing-agents predominate, cyanide will not dissolve the gold from the ore. In many cases it will dissolve some of the gold, because in a mass of irregular shape some of the gold particles might be exposed upon the outside surface of a particle of rock; but if the solution had to penetrate through cracks, the side-walls of which were lined with reducing-agent-

producing material, before the solution carrying oxygen could reach the gold it would have lost its oxidizing power. For this reason in many cases cyanide solutions will produce only a partial extraction of the gold or silver present.

It occurred to me that since water is composed of hydrogen and oxygen, if it be decomposed by the electric current, the hydrogen would bubble away and the oxygen would be carried by the solution. This was tried in December, 1908, upon an ore carrying amorphous iron sulphides from which all the gold could not be dissolved by cyanide with simple air-agitation, no matter what the cyanide strength or how great the time, although the gold as revealed by the microscope was all metallic. The process was tried first in an inverted bottle with the bottom cut out, the air being forced in through a glass tube in the cork to agitate the pulp. Two lead plates were inserted in the agitated pulp at the top. These plates were about 4 in. long and 0.5 in. wide, and $\frac{1}{16}$ in. thick. Through them was passed the current of an incandescent lamp, which being in series and burning dimly gave about 0.25 ampere of current. The results were excellent from the beginning. The value of the ore was $4 per ton. It was ground in a tube-mill so that 60 per cent. passed a 200-mesh screen. The value of the tailings, after 48 hr. agitation with air alone, was $1.25; but after agitation for 2.5 hr. with air and electrodes inserted in the pulp as described above, the value was reduced to $0.40. This typical result was verified perhaps a thousand times, with uniformly good results.

In testing our solutions, a 2-lb. solution of cyanide is test 10. The alkali is tested on the basis of ten points over and above the alkali due to the cyanide, test 10 being a 2-lb. solution of caustic soda. The reducing-agents were tested with a 1 per cent. solution of potassium permanganate, 1 cc. of which in 10 cc. of the solution, after acidulating, equals test 10, it being much easier to keep track of these solutions by simple numbers than by keeping the records in pounds per ton.

Numerous tests were made in order to determine a proper electrode. Lead was found to be the best material. Many other substances, such as carbon, worked very well, but with the alternating current, there being no consumption of the lead electrode, lead proved most satisfactory.

The following tests upon the working-solution show the effect of the different electrodes. All the tests were made at the same time and with the same solution, using the direct current.

Lead electrode: Time, 4 min.; 0.25 ampere current.

	KCN.	Alkalinity.	Dbl.	Reducing-Agents
Before. . .	8	+ 1	5	6
After, . .	12	+ 2	0	4

Iron electrode: Time, 6 min.; 0.25 ampere current.

	KCN.	Alkalinity.	Dbl.	Reducing-Agents.
Before, . .	8	+	5	6
After, . . .	7½	+ 6½	0	3

Iron electrode: Time, 12 min.; 0.25 ampere current.

KCN.	Alkalinity.	Dbl.	Reducing-Agents.
6	7½	0	2

(Showing destruction of the cyanide.)

Lead electrode: Time, 10 min.; 0.25 ampere current.

KCN	Alkalinity.	Dbl.	Reducing-Agents.
12	3		3

There seems to be a regeneration of cyanide, and the process is certainly cheaper than any added oxidizer or even air-agitation of the solution.

We found by numerous experiments that the alternating current was as good as the direct current, and had the additional advantage of giving no deposit on the electrodes at lower current-density, and with lead there was no consumption of the electrodes even where the ore-pulp flowed over the electrodes. The way I explain this result is as follows:

Under the prevailing conditions certain electro-chemical actions take place by which the particles composing a molecule of a compound are resolved into the parts that the applied current-strength would resolve them into, and go into the solution on the one wave, and they do not re-combine on the returning current wave. In other words, dissociation takes place without being followed by re-combination. At any rate, no matter how the action is explained, it is carried on and works satisfactorily.

In electroplating, if the current is of low density the material deposited will be dense. If the current-density is increased,

the material deposited will be spongy. If the current-density is still further increased, the material which should be deposited will be disengaged by the action of the gases, and practically no deposit will result, the material going into the solution in a more or less spongy condition. We found that with a very high current-density no deposit of gold or silver accumulated upon the lead electrodes with direct current. Some of the electrodes after being in use six months were scraped, and the scrapings assayed, and showed only a trace of gold and silver.

Electrolyzed solution seems to be especially effective when used in connection with lead acetate or litharge added in the tube-mill during grinding. The electrolyzed solution going to the tanks shows no sulphocyanides, whereas, before the batteries were put in use, the solution showed a large amount.

As finally used in practice in January, 1909, a battery, supplied with alternating current, was placed in the barren sump. This battery consisted of 18 plates in series, each plate 6 by 6 in., with 110 volts between the two. The plates consumed 15 amperes, and produced sufficient oxidizing effect, or whatever other effect it may be, to keep the solution in condition to treat daily 40 tons of this ore. These plates, made of $\frac{1}{8}$-in. sheet-lead, were built so as to form hollow rectangles in section, the rectangle being 6 in. high, 6 in. long, and 1.25 in. wide inside. The two ends were lapped at the top and holes punched. The plate was bolted to a paraffined plank 1 by 6 in. in section; 18 of these plates were connected in series. The distance between any two plates was $\frac{1}{8}$ in., and, of course, the current would travel principally across the $\frac{1}{8}$-in. gap, instead of around the $1\frac{3}{8}$-in. gap, from plate to plate. Lead wires were used from the surface of the solution down to the plates. We ground the ore in the tube-mill so that 60 per cent. would pass a 200-mesh sieve. Previous to using the batteries in the sump, the extraction in the tube-mill was 20 per cent. during grinding; after the batteries were used, the extraction in the tube-mill was 75 per cent. The effect of the batteries seemed to build up in the solution gradually and to lose from the solution gradually when the operation of the batteries was discontinued.

During two months in the fall of 1910 the mill was working coarse ground, partly-oxidized ore carrying considerable sul-

phides. The water at the hydro-electric plant was low, and the use of the batteries was discontinued because the mill was driven with steam, and no arrangement had been made to supply alternating current from any but the hydro-electric plant. During this time the tailings on \$4 ore went up to \$1.25 per ton, and immediately after the rains gave sufficient water to drive the hydro-electric plant, the values in the tailings diminished until \$0.20 per ton was reached on identically the same ore with the same head-values; moreover, the reducing-agents dropped from 16 to 4. The time occupied in getting the working-solution up to this condition was two weeks. I consider that this process owes its value almost entirely to the presence of oxygen due to electrolysis, putting the solution ahead in the race with the reducing-agents and causing the gold and silver to dissolve in spite of the reducing-agents. However, it does not stop the reducing-agents from dissolving also, and although it produces solution of the gold in spite of the reducing-agents, it does not help precipitation, and if the reducing-agents are not decomposed by the batteries—and all of them are not—they build up in the solution rapidly to a point where zinc-shavings will not precipitate the gold.

Of course, in practice the cyanide solution contains reducing-agents of many kinds. The electrolytic action seems to reduce the influence of some, but not all of them. For example, I experimented on some highly-graphitic ore, and whether the normally-poor extraction was due entirely to the graphite or not, I do not know; but the solution, after electrolyzing, gave a very much better extraction than before electrolyzing. The action seems to decompose the sulphocyanides and the soluble sulphides, but not the alkaline sulphides and all of the many others always present.

A test on the electrolyzed solution 18 months after the batteries were installed showed:

Working-solution with alternating current, 0.25 ampere, and lead electrode.

	KCN.	Alkalinity.	Dbl.	Reducing-Agents
Before, ...	8	1	0	15
After 10 min. electrolysis,	8		0	15

showing that the solution remained practically the same, or was electrolyzed as much as was necessary. However, testing

some of this same solution further by placing a piece of gold leaf upon its surface and allowing it to float, the gold leaf was dissolved in 71 min. on the working-solution and in 50 min. on the re-electrolyzed solution, showing that the additional electrolysis, although it had no apparent effect on the solution, gave an increased dissolving-rate. Grease in the ore or on the surface of the barren sump seemed to dissolve very rapidly in the treated solution and slowly in the untreated solution. We tarred our tanks inside and coated them with black oil outside, and more or less grease was frequently floating upon the surface of the solution where this effect was noticed.

Since the installation of this process it has treated successfully at this plant 25,000 tons of ore of all kinds, oxidized, partly oxidixed, and sulphides. Previous to the use of the batteries, in treating sulphide ores, the average cyanide-consumption was 1 lb. per ton, in some months running as high as 1.1 lb. After the use of the batteries the average was 0.45 lb., running for some months as low as 0.23 lb. per ton of ore treated.

We tried using batteries in the agitated pulp and in the solution, and found the result to be just as good if the plates were inserted in the barren sump as if inserted in the agitated pulp. The original lead plates placed in the barren sump are still there and in operation. They cost about $4 to insert originally and were inspected after 26 months of practically continuous service, and are to-day just as good as when they were first put in use.

I have applied for no patents on this process and do not expect to, and any one is free to use it. It should be a cyanide-saver, an accelerator, and a general solution-purifier.

Slime-Filtration.

BY GEORGE J. YOUNG,* RENO, NEV.

(San Francisco Meeting, October, 1911.)

THE nature of slimes handled in the treatment of gold- and silver-ores has been discussed in technical literature to a considerable extent. The subject of slime-filtration from the practical worker's stand-point has also received much comment, and scattered through the literature of the subject are descriptions of many slime-filtration installations. Articles of this nature serve a valuable purpose and assist materially in the design of new and the improvement of old plants. The subject of the physics of slime-filtration has been touched upon to only a slight extent. The underlying principles are worthy of more intensive study and experimentation than they have received, and the main purpose of this paper is to present the results of such study and experimental work as will serve to make clear in part at least many of the principles which control the filtration of slime.

Nature of Slime.

Much has been written concerning an accurate definition of the term "slime," but no comprehensive definition seems to be generally accepted. The reason for this is clear. A slime consists of at least three different substances, each, when separated, possessing distinctly different physical and to a certain extent chemical properties. These substances are extremely fine sand, a colloidal material which may be and generally is in a coagulated condition, and a colloidal material which is in a non-coagulated condition. Suspending a slime in a relatively large volume of water by shaking and allowing sedimentation to take place, results in the fine sand settling out with comparative rapidity, followed by the coagulated material, which settles much more slowly and finally a certain portion remains

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indefinitely suspended. To the settled, coagulated portion some writers have given the term *gel*, and to the suspended portion the term *sol*. The physical properties which distinguish the fine sands are, the angular character of the grains and the comparatively rapid settling in water. With most quartzose ores the sand grains are composed of silica, although constituents of the ore, such as silicates or oxides, also characterize the sand portion of the slime. The coagulated colloid consists of aggregates of rounded grains together with individual grains, settles much less rapidly, possesses the property of flocculation and deflocculation, has the property of absorbing certain dyes, and a distinctive chemical composition. Clays and hydrated silica are the two colloids most likely to occur in quartzose ore. The former is a common constituent of many ores, the latter is perhaps seldom present. For practical purposes, clay, or hydrated aluminum silicate, may be considered to be the chief colloidal constituent, and, mixed with fine sand, to constitute the coagulated portion of a slime. Inasmuch as the ordinary mill-slime is quite well coagulated by the liberal use of lime, the metallurgist has to deal only with mixtures of fine sands and coagulated colloid.

The distinctive properties of a slime depend upon the relative proportions of fine sand and colloid. Assuming all colloid and no fine sand, we would have a material which could not be leached, and which would filter very slowly, and under certain conditions not at all; assuming all fine sand and no colloid, we would have a leachable material. In a moist condition a slime may be likened to a clay; with a large proportion of sand a "short clay" or a clay of moderate plasticity would be the result; with a small proportion of sand a "fat" clay or a clay with a high degree of plasticity would result. With sufficient moisture a slime partakes of the character of a viscous fluid, and in this very fine sand will be almost indefinitely suspended, and little or no separation of fine sand from colloid will result. This latter statement is true of sand finer than a 150-mesh screen. With coarser sand, the coarse sand particles tend to settle out quite rapidly. By increasing the proportion of water successive crops of finer and finer sands can be settled out until a point is reached where the particles of coagulated colloid and the finest sands settle at the same rate. Beyond

this point no further separation of sand from colloid is possible. No sharp line in the mechanical separation of sand from colloidal material being possible, it is necessary to use a definition which will embody some limitation as to the size of the maximum sand grain. Successive screen-sizes have been used; first a 100-mesh screen, then a 150-mesh screen, and, finally, a 200-mesh screen; and this is the accepted present practice in milling-work. All material in a pulp finer than a 200-mesh screen is considered as slime. The definition, that a slime is the unleachable portion of a mill-pulp, is still in use.

A more comprehensive definition than the foregoing is: a slime consists of a mixture of sands finer than 150- or 200-mesh screen with an amorphous clay-like material, consisting principally of hydrated aluminum silicate.

The general method of slime-treatment is to agitate the slime with a cyanide solution for a sufficient time to dissolve the gold, and then, either to filter off the surplus solution and displace the remainder with water, or to thicken the slime by settlement and decantation, and then to filter and displace the remaining solution by water.

The mechanical appliances in use for filtration are grouped as follows:

I. Suction-filters, or filters in which a vacuum is used to accelerate filtration.

A. Appliances using a thin slime-cake and practically continuous in their action. (Oliver and Ridgway filters.)

B. Appliances using a thick slime-cake and intermittent in their action. (Moore and Butters filters.)

II Pressure-filters, or filters in which hydrostatic head, compressed air or pumps are used in order to secure greater pressures than are possible with a vacuum-pump.

These filters are intermittent in their action.

C. Ordinary filter-presses.

D. Sluicing filter-presses (Merrill filter-press.)

E. Filtering-chambers or cylinders; filters in which the filtering-basket is inclosed in a cylinder. (Burt, Kelly, and Sweetland filter-presses.)

III. Centrifugal filters, or filters in which centrifugal force is used to separate solution from slime.

These filters are continuous in action.

The filters in Sections I. and II., with the exception of the Ridgway, employ vertical filtering-surfaces. The Oliver[1] makes

[1] *Trans.*, xli, 349 to 356 (1911).

use of a revolving cylindrical surface as a filtering-surface. Centrifugal filters are in process of development, and have not as yet secured any foothold in gold- and silver-metallurgy. It is not improbable, however, that some comparatively simple filter based on the use of centrifugal force will be perfected, and will successfully compete with the other forms. At present the suction-filters are in greatest use. Of the pressure-filters, the ordinary filter-presses have gone out of use, except as clarifying-presses, and filters of groups *D* and *E* only are in use.

The development of slime-filtration is of interest. Filter-presses and filtering-beds in vats were first used. The filtering-beds were soon discarded and the filter-press systematically developed. The size of the press was increased, mechanical devices to facilitate discharge and decrease the proportion of labor required were invented and introduced; but in spite of all this the cost of treatment in filter-presses remained high. In western America the filter-press never received much recognition, but in Australia filter-pressing was extensively introduced, and slime was successfully handled by this method. It remained for an American, Charles A. Merrill, to complete the last improvement in the filter-press. By the introduction of the sluicing-system the slime-cakes could be washed out of the filter-cells and the press operated without opening or separating the filter-plates for each charge. This improvement reduced the labor and cost and increased the effectiveness of the filter-press. The Merrill press represents the culminating point in the filter-press line of development in slime-filtration.

The Moore filter was the first suction-filter in the field, and, while it did not score any very decided success in the first installations, it did attract the attention of metallurgists to the idea involved. While the Moore Filter Co. was perfecting the mechanical features of its filter, the Butters filter was introduced, and so many of the difficulties of the Moore filter were overcome in the Butters, that this latter filter received widespread recognition and was introduced into many milling-plants. The Moore filter introduced the idea of the canvas-covered filtering-cell immersed in the slime-pulp and utilizing suction to draw the solution through the walls of the cell and to build up a cake. The necessary transfers are made by lift-

ing the filtering-basket out of the pulp. The Butters filter introduced the idea of a stationary filtering-cell, and effected the transfers by pumping the slime-pulp and wash-water from the vat in which the filtering-cells were immersed. The relative merits of the two systems have been sufficiently discussed in the technical literature. Both the Moore and the Butters filter have reached a point where little or no further improvement seems possible. Like the Merrill, either one of these systems will satisfactorily meet the requirements of slime-filtration.

The combination of the ideas involved in the filter-press and the suction-filter is seen in group *E*, or the filtering-chambers. The Kelly, the Burt, and the Sweetland may be compared to a Butters filter installed in a pressure-tank.

The effort to secure a continuously-acting filter has resulted in two important types being developed, of which the Ridgway and the Oliver are the best known. Both of these filters utilize a comparatively thin slime-cake. Both operate very successfully, and compared with the thick-cake machines have decided advantages, briefly stated as: simplicity of design; probably lower capitalization-charges for equal capacities; lower operating-costs; and less attention required in the operation.

With the exception of the Oliver filter, the general method of operation of both suction- and pressure-filters is the same. The slime-pulp is delivered to the filter in the proportion of one of dry slime to from three to one of solution. The pulp is forced into the cells of the pressure-filters and a cake formed against the canvas walls of the cells, the surplus pulp, if any, is withdrawn, and wash-water forced in until the contained solutions are displaced. The cake is then forced off from the canvas surface, either by water or air or a combination of both, and sluiced out. In the vacuum-filter the filtering-cells are immersed in the pulp, a vacuum is formed, and a cake built up; the surplus pulp is then withdrawn either by lifting the filtering-cells out or by withdrawing the pulp by pumps, and the cakes are immersed in water for washing. In the Moore filter the cakes are discharged by forcing them off from the cell by water or air and dropping into a hopper for sluicing away; in the Butters the cake is forced off in the same way, but while still immersed in the wash-solution. The wash-solution is then

withdrawn, either by decanting or pumping, and the slime-cake and surplus wash sluiced out. The Oliver filter performs the operations of cake-formation, washing, and discharge in continuous sequence. Three steps may be designated as common to all these filters: cake-formation, washing, and discharge. The cycle of operations of the more common forms of filters is shown in Fig. 1. Typical examples have been taken in each case.

The conditions under which slime-cakes are formed and washed are the critical points to be considered; the discharge and sluicing away of the cake is a comparatively simple mat-

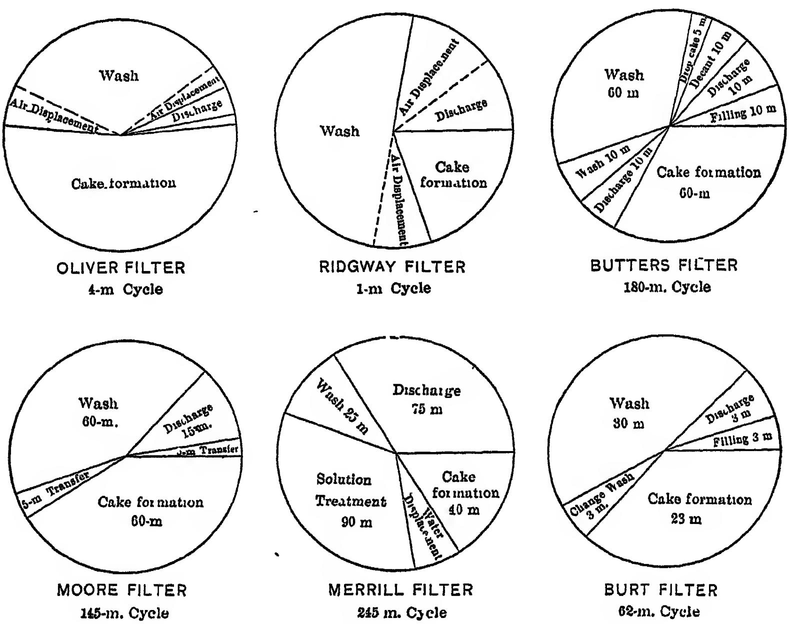

FIG. 1.—CYCLE OF OPERATION OF VARIOUS FILTERS.

ter and requires no special comment. My experimental work was largely confined to suction-filtration, and pressure-filtration was only briefly studied. The method of carrying out the experiments may be summarized as follows: After trying out several different sizes and types of filter-cells a test-filter of 0.5 sq. ft. filtering-surface was decided upon, shown in Fig. 2. A ribbed wooden support with $\frac{1}{8}$-in. grooves and $\frac{1}{8}$-in. ribs was used to support the canvas surface. Brass side-strips and a

slotted brass bottom-strip were used to protect the cake and to assist in measuring. A type slime was obtained by classifying a pulp from a Tonopah quartzose ore which had been crushed in a stamp-battery. The slime was settled by the use of lime, and then by repeated settlement all the coarse and as much of the fine sand as possible were settled out and removed.

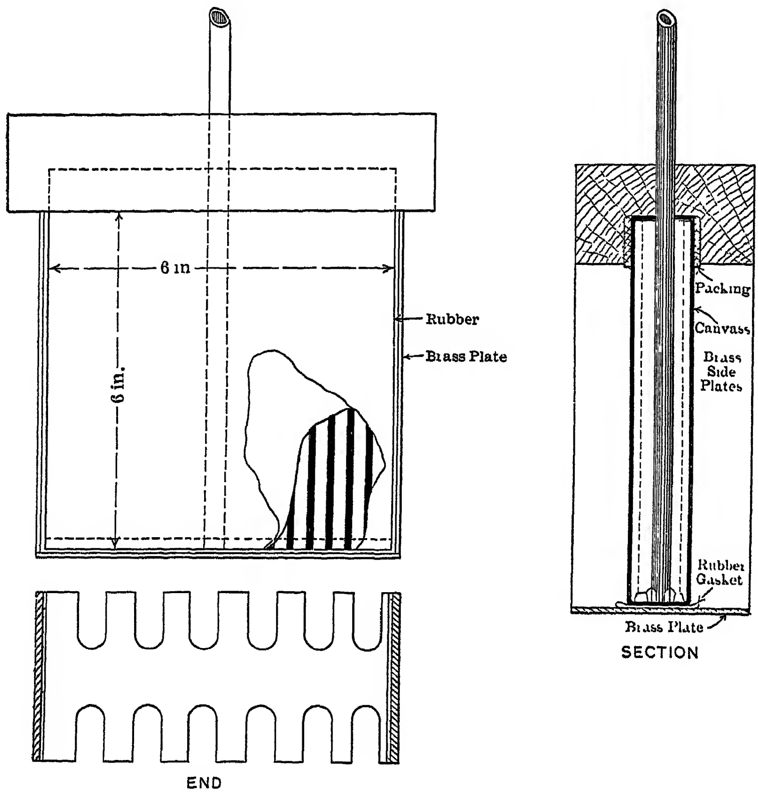

FIG. 2.—FILTER USED IN EXPERIMENTAL WORK.

The slime-pulp remaining was settled to a thickness giving a density of 1.3. The screen-analysis of this slime approximated:

	Per Cent.
On 100-mesh,	0.1
Plus 150, minus 100,	1.3
Plus 200, minus 150,	1.2
Minus 200, and less than 2 min. settling,	20.0
Settling in from 2 to 4 min.,	8.5
Settling in from 4 to 8 min.,	11.6
Remainder,	54.3

Of this slime-pulp, 97.4 per cent. passed a 200-mesh screen. Fine sand passing a 100-mesh screen was used in securing the necessary mixtures. The filtrate was measured in a Woulfe bottle, to which was attached a vacuum-gauge. The vacuum was obtained by a small single-acting pump exhausting from a 10-gal. vacuum-tank. A short length of hose connected the Woulfe bottle with the tank. The slime-mixtures were made up in buckets and heated to the temperatures as required. Variations in pressure, temperature, and slime were the main points studied.

Filtering-Rate.

The rapidity with which a cake may be formed depends upon the filtering-rate of the slime, the thickness of the cake, the temperature and density of the pulp, and the intensity of the vacuum. The filtering-rate of a slime, which is numerically defined in this paper as the number of pounds of water drawn through 100 sq. ft. of filtering-surface per minute, depends, for a cake of given thickness, upon the character of the slime, the density of the slime-cake, the suction-pressure, the temperature, and, to a moderate extent, upon the character of the filtering-surface and its support. These factors are so interrelated that it is impossible to conduct any series of experiments which would exactly show the effect of varying them. At best, the results are approximations.

Fig. 3 shows the variation of the filtering-rate with variable thickness of slime-cake both while building up and in clear water. The curves represent the averages of a number of tests in which temperature and pressure were practically the same for all. A No. 10 canvas was used. In carrying out the experiment the filter was immersed in the pulp for 5 or 10 min. and a cake built up. This cake was then quickly removed, its thickness measured, and the filter immersed in clear water. After determining the filtering-rate, the filter was replaced in the pulp and an additional thickness built up. The filtering-rate during building up was determined by calculation from the amount of water passing while building to a given thickness. The difference between the two curves is comparatively slight and indicates that the filtering-rate during building up a cake is greater in the pulp than in clear water for thin cakes, while for the thicker cakes the reverse is true.

Fig. 4 shows the effect of variation of pressure upon the filtering-rates of cakes of varying thickness. Three pressures were used—11.35, 17, and 21.5 in. of mercury. The last pressure is about the maximum obtainable in Nevada practice. The general effect of increase of pressure is to increase the filtering-rate. This is more marked with the thin cakes, while with the thick cakes all three curves tend to run together. With thick cakes the effect of an increase of pressure is to increase the density of the cake and thus reduce its permeability. With higher pressures this effect is more marked, and indi-

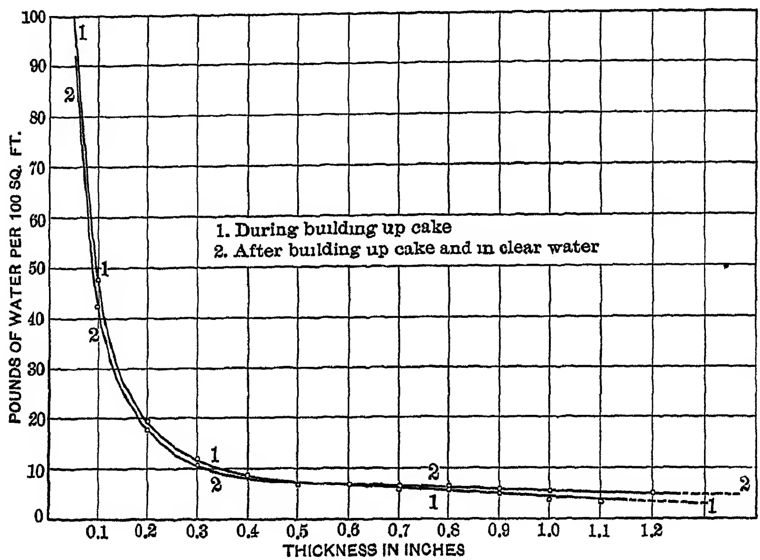

FIG. 3.—AVERAGE FILTERING-RATES FOR SLIME-CAKES, NO. 10 CANVAS.

cates that a point would soon be reached where the increased pressure would result in decreased filtering-rate. This is particularly true of slime containing a small proportion of sand, and much less so with slimes containing a large proportion of sand. Sweetland, in his paper, Pressure Filtration,[2] shows for pressures up to 65 lb. per sq. in. a progressive increase in the filtering-rate for slime-cakes varying from 0.5 to 1.75 in. The slime used in the Sweetland experiments was obtained from the Goldfield Consolidated mill. Unfortunately, neither a physical analysis of the slime nor the density of the slime-cakes formed

[2] *Mining and Scientific Press*, vol. xcix., No. 26, p. 853 (Dec. 25, 1909).

is given in the paper. The slime-pulp of the Goldfield Consolidated mill is distinctly of a sandy nature and would be expected to give results of this kind, whereas a very clayey pulp would give results of an opposite character. Experiments with a slime similar to the type slime, and with pressures ranging from 10 to 30 lb. per sq. in., showed an increase in filtering-rate from 11 to 16 lb. of water per 100 sq. ft. per min. for a cake of 0.25 in. thick; for a 0.5-in. cake an increase in pressure from 20 to 30 lb. decreased the filtering-rate from 10 to 7 lb.; for a 0.75-in. cake an increase in the filtering-pressure from 20 to 30 lb. made no difference in a filtering-rate of 6 lb. R. Gilman Brown, in his paper, Cyanide Practice with the Moore Filter,[3] in discussing the treatment of a very clayey

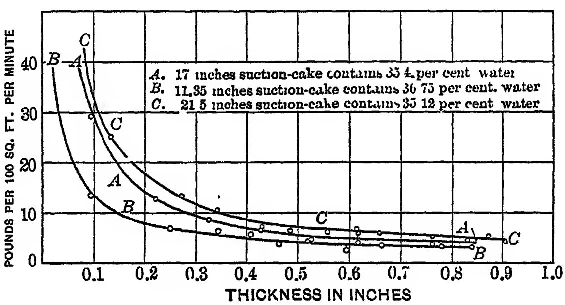

FIG. 4.—EFFECT OF VARIATION OF PRESSURE UPON FILTERING-RATES, NO. 10 CANVAS.

slime at Bodie, says: "Filter-pressing was tried and abandoned, because an eighth of an inch of pure slime would make the cloths impervious, even under 120-lb. pressure; and even if the slime was mixed with fine sand, the filtering was so slow that the sand settled out in the chambers, with the same result." The practical conclusion that may be drawn from a study of the effects of pressure in filtration is that, with material of a permeable nature such as a sandy slime, increased pressures over those obtainable by means of vacuum-pumps are advantageous, while with material in which only a moderate to a small amount of sand is present and the permeability low, the use of higher pressures offers no advantages over those obtainable by vacuum-pumps. In the use of both the Moore and

[3] *Mining and Scientific Press*, vol. xciii., No. 9, p. 261 (Sept. 1, 1906).

the Butters systems, experiments should be made with different intensities of vacuum, for it may be found that a vacuum lower than the maximum obtainable with the available apparatus will give a higher filtration-rate, and thus decrease the time for both building up and washing.

Fig. 5 gives the comparative filtering-rates of five slimes. The same test-filter, temperatures, and pressures were used in each case. No. 10 canvas was used on the filter. The slimes used were: a clay slime (a very plastic fire-clay) containing about 40 per cent. of sand which settled out in 1 min.; the average of the results on the type slime; a slime from a Virginia City tailings-pond; the type slime containing 37 per

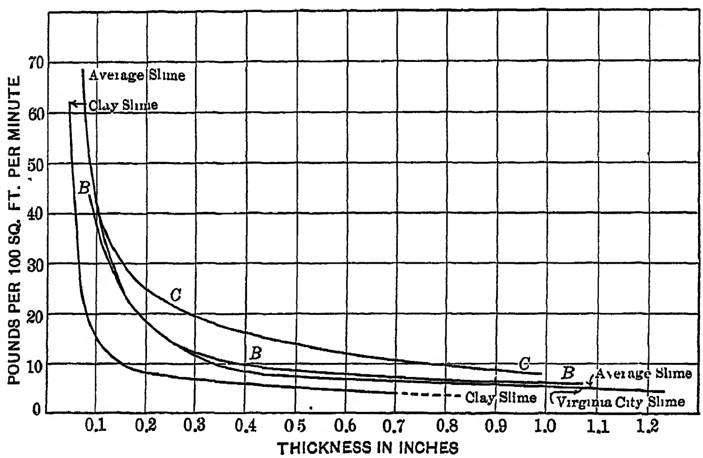

FIG. 5.—FILTERING-RATES OF FIVE SLIMES, No. 10 CANVAS.

cent. of fine sand (determined on the basis of 1 min. settlement); the type slime with 52 per cent. of fine sand (determined on the basis of 1 min. settlement). The type slime on the basis of 1-min. settlement gave 6.5 per cent. of fine sand. *B* and *C* respectively represent the 37 and the 52 per cent. of fine-sand slimes.

The filtering-rate curves for the type slime and the Virginia City slime are coincident. The increase in the proportion of fine sand from 6.5 to 37 per cent. makes but very little difference in the filtration-rate. A further increase to 52 per cent. shows a marked increase in the filtering-rate (curve *C*). While the clay slime has a greater proportion of fine sand than either

the type or *B*, the filtering-rate curve is much lower. The conclusions which may be drawn from these experiments are: slimes from similar ores subjected to the same metallurgical treatment give similar filtering-rate curves; a moderate variation in the proportion of fine sand gives filtering-rates differing

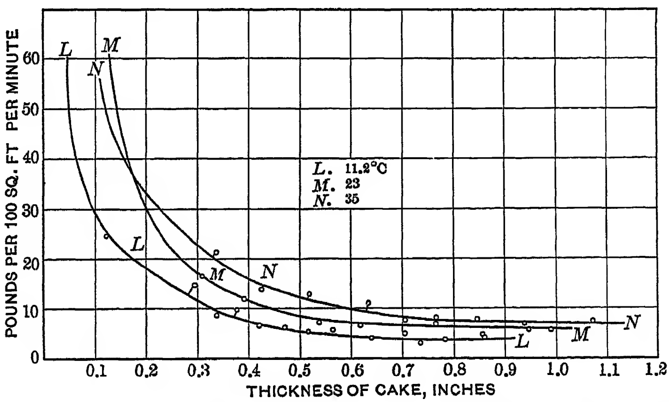

FIG. 6.—EFFECT OF TEMPERATURE UPON FILTERING-RATES OF SLIME-CAKES, NO. 12 DUCK.

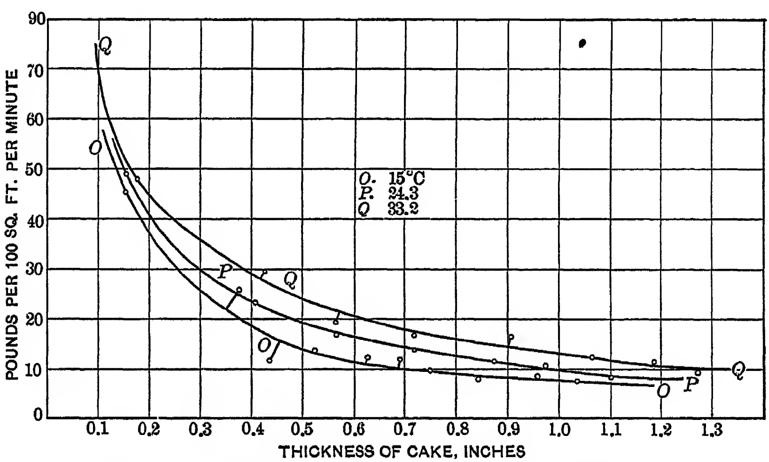

FIG. 7.—EFFECT OF TEMPERATURE UPON FILTERING-RATES OF SAND- AND SLIME-CAKES, NO. 12 DUCK.

only to a small degree, while a considerable increase in the proportion of fine sand increases the filtering-rate; the proportion of colloidal matter, or, in this case, clay base, has a marked influence upon the filtering-rate; much more, relatively, than

the effect of fine sands in increasing the filtering-rates. The amount of clay is the dominating factor in filtering-rates, and this fact is indicated by the curves approaching a common point as the thickness of the cake is increased.

Fig. 6 shows the effect of temperature upon the filtering-rate of the type slime. A No. 12 canvas was used on the filter for these experiments. For thin cakes the increase in filtering-rate is more marked than for the thicker cakes. The same tendency of the rate-curves to run together for the thicker cakes is to be noted.

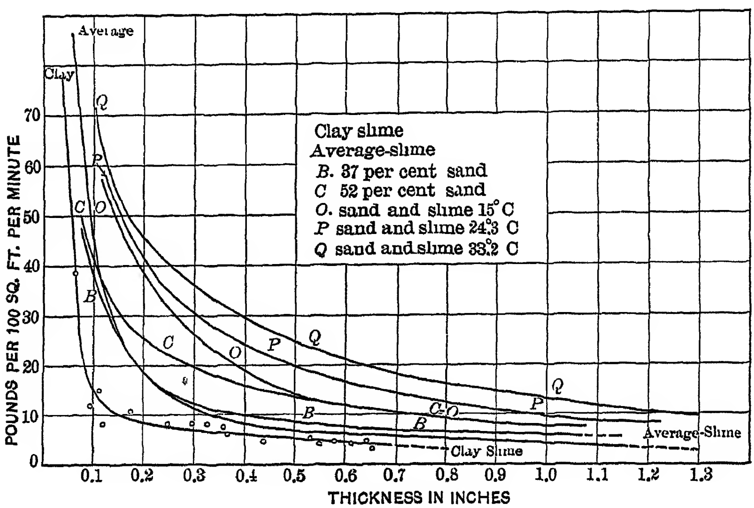

FIG. 8.—COMPARISON OF FILTERING-RATES, NO. 12 CANVAS.

Fig. 7 shows the effect of temperature upon a slime containing 50 per cent. of fine sand and the type slime. The marked increase in filtering-rate with moderately elevated temperatures is so noticeable as to indicate a condition of considerable practical importance. By increasing the temperature of the pulp greater capacity could be readily obtained with a given unit. Fig. 8 compares the filtration-rates of the clay slime, the type slime, and the several sand-slime cakes. Fig. 9 compares the filtering-rates for fine-sand beds 3 in. thick under varying pressures.

Filtering-Surfaces.

Most of the suction-filters employ No. 10 canvas duck for the filtering-surface. The Oliver filter makes use of a No. 12

and the Merrill filter-press of a No. 6 duck over a light twill. In the Butters and the Moore filters three methods of support are in common use. The original Butters unit consisted of canvas stitched at close intervals over a center sheet of cocoa matting, which gives a very porous gathering-space for the solutions and also sufficient support to the canvas. The objections to this construction are the cost, and the clogging of the matting. With the exception of the Goldfield Consolidated mill, all the mills in the Tonopah and Goldfield districts employ the "slat method" of support, which consists of sewing the canvas walls of the cell into narrow pockets from 1.5 to 2 in. wide, and into each of these slipping a grooved lath. The arrangement is low in first-cost and very satisfactory. The Moore system employs

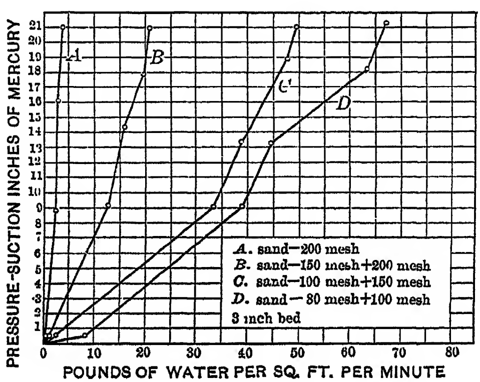

FIG. 9.—FILTERING-RATES FOR FINE-SAND BEDS.

wooden strips slipped into narrow pockets in the canvas. The Moore system also makes use of wire netting between the canvas walls, the canvas being stitched at frequent intervals through the netting. In the Oliver filter, wire netting over a grooved board and covered with 8-oz. burlap supports the canvas. The canvas is held against this base by wire wrapped around the canvas at 0.5-in. intervals. In both the Butters and the Moore filters wooden dividing-strips are used to space the filtering-surface into strips 1 ft. wide. Grooved iron plates are used in the filter-presses and in the Merrill press.

Durability and permeability are the necessary requirements of a filtering-cloth. Canvas duck, army weave, No. 10, answers both of these requirements for suction-filters. For pressure-

filters this canvas is too light, and No. 6 gives sufficient durability without interfering with the filtration too much. On account of the wire wrapping, the Oliver filter can employ a lighter duck (No. 12). The relative permeability of different weights of canvas is a difficult matter to determine experimentally. It is a function of the weave of the cloth and the character of the supporting surface. In general, the lighter the weight of the duck the more permeable it is. Duck may be obtained in three weaves: army duck, in which the threads of warp and woof are twisted; double-fill, in which the warp thread is twisted and the woof threads are not twisted; single-fill, in which neither the warp nor the woof threads are twisted. Of

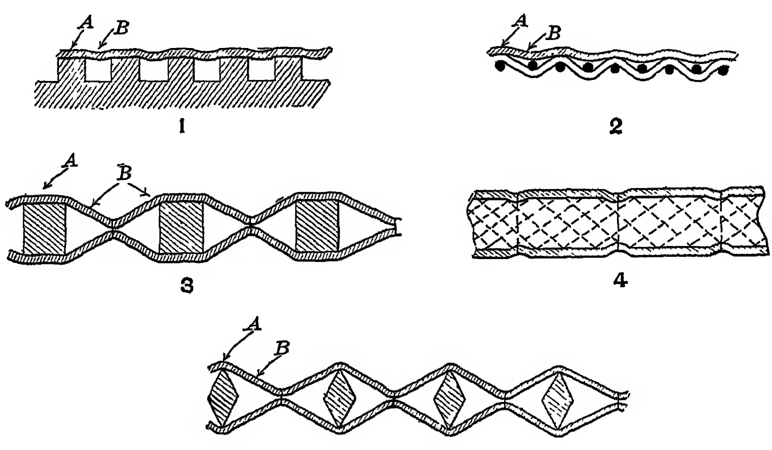

FIG. 10.—METHODS OF SUPPORTING FILTER-CLOTH.

the three weaves the army duck is the least permeable, while the other two weaves are too open and porous to be of much use in slime-filtration.

Fig. 10 illustrates several methods of supporting the filtering-cloth. In method 1, the fibers of the cloth are distended and the cloth made more open at *B*, while at *A* the fibers are flattened and pressed against one another, with the result of reducing the permeability of the cloth over the ridges. The relative proportion of ridge to groove determines the decrease in permeability due to the support. No. 2 shows the wire netting support, and with this the rounded wires reduce the permeability to a less extent than the flat wooden ridges. In No.

3 the narrow wooden strips have less effect than the close ridges of No. 1, while between the strips the cloth is stretched and is more open. With No. 4 the permeability of the cloth is not generally affected, for the fibers may press into the soft cocoa matting. In No. 5 the diamond-shaped strips give the maximum proportion of distended canvas, and leave the fibers free from any flattening due to the pressure.

Fig. 11 shows the effect upon the filtering-rate of the type slime for three filters. Curve *L* is for No. 12 duck on a grooved wooden support similar to No. 1, Fig. 10; 41 per cent. of the cloth was supported and 59 per cent. unsupported. Curve *S* is for No. 12 duck supported on diamond-shaped

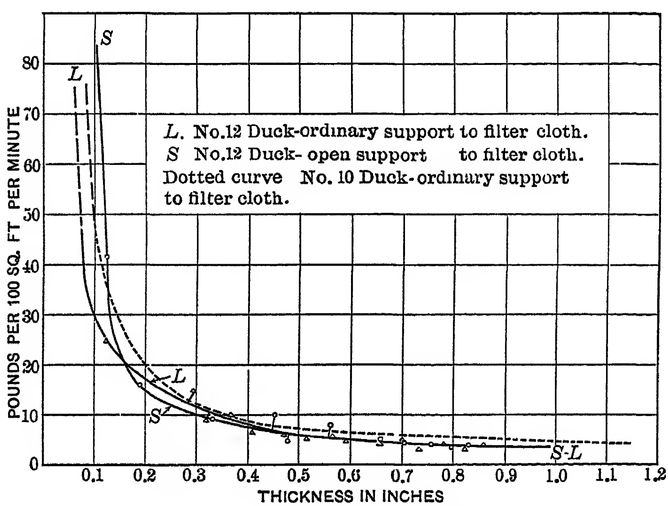

FIG. 11.—FILTERING-RATE OF TYPE SLIME FOR THREE TYPES OF FILTERS.

strips; 21 per cent. of the cloth was supported, 79 per cent. unsupported. The dotted curve is the average curve for the test-filter. A No. 10 duck was used and the same proportion of support given the cloth as for curve *L*. The heavier-weight duck on the grooved support gives a higher filtering-rate than the light weight on either the grooved or the more open support. The open support gives a higher filtering-rate for the thinner slime-cakes and lower rates for the thicker cakes. This anomalous result is explained by the fact that the more permeable the filter the more active becomes the filtering-surface for a given pressure and the more compactly the cake

is built up. The general conclusion is that the permeability of the filter-cloth is a matter of moderate importance; of greater importance with the thin-cake suction-filters than with the thick-cake filters.

No experiments could be made on the comparative durability of filter-cloths. From data submitted in Table V. it appears that suction-filters, supported with cocoa matting, have the longest life. Close stitching is of importance both in maintaining an efficient filter and in prolonging the life of the filter-cloth. The first Moore filters were constructed with the canvas supported at 6-in. intervals, and these filters failed by tearing and weakening generally at the points of attachment. With stitching at 1-in. intervals the wear at the stitching-points is reduced very materially in the Butters filter. With the slat-filter narrow slats allow closer stitching, and this is the tendency in construction.

The grooved support results in practically a clear solution from the start of filtration whether a No. 10 or a No. 12 duck is used. The diamond-strip support and No. 12 canvas gave a turbid filtrate for the first minute of filtration.

The relation between numbered duck and ounce duck is as follows:

Numbered Duck.	Ounce Duck 29 In. Wide and 36 In. Long.	Ounces Per Square Yard.
	Ounces.	
No. 12	8	10
No. 10	12	14—15
No. 8	15	18—19
No. 6	18	22—23

Slime-Cakes.

A slime-cake is built up of a succession of thin layers of slime. Slices taken from the surface, middle, and next the canvas showed varying percentages of water, and consequently a variation in density from the outer surface to the canvas. Fig. 12 represents graphically the results of sectioning different slime-cakes. *B* shows the proportion of sand, slime, and water for a cake made from the type slime. The cake was 0.81 in. thick, and the vacuum used 21 in. The outer 0.25 in. contained 41.4; the middle, 39.6; and the portion

next the canvas, 33.3 per cent. of water. The respective specific gravities are 1.575, 1.605, and 1.715. The average water-content is 38.1 per cent., and the average specific gravity, 1.632. *F* and *G* are from two slime-cakes in which the respective proportions of sand were 37 and 52 per cent. The composition of sections of these cakes is given in Table I.

TABLE I.—*Composition of Slime-Cakes.*

CAKE F.

	Surface 1 to 0.75 In.	0.75 to 0.5 In.	0.5 to 0.25 In.	Next Canvas 0.25 to 0.0 In.
Specific gravity of cake, .	1.755	1.805	1.822	1.835
Per cent. of water, . .	32.7	29.83	28.41	27.9
Per cent. of sand in dried cake, ...	36.02	35.23	37.82	38.04
Ratio sand to slime, 1 to, .	1.77	1.83	1.64	1.62

Average sp. gr., 1.804. Average percentage of water, 29.71.

CAKE G.

Specific gravity of cake, .	1.776	1.814	1.853	1.841
Per cent. of water, . .	30.0	28.4	26.9	27.4
Per cent. of sand in dried cake, ...	51.6	54.0	53.7	50.1
Ratio sand to slime, 1 to, .	1.08	1.10	1.17	1.22

Average sp. gr., 1.821. Average percentage of water, 28.4.

Comparing the results in Table I. with those given for the type slime, a variation in water-content of from 38.1 to 28.4 per cent., and in density of from 1.632 to 1.821, is shown for a variation in sand of from 6 to 52 per cent. The sand in these cakes was determined on the basis of 1-min. settlement. An increase in the proportion of fine sand decreases the interstitial space, but does not decrease the permeability, as the curves for filtering-rates show. For purposes of comparison, the proportion of water absorbed by fine sands is shown in Fig. 12. Fine sands (quartz) between 80- and 100-mesh contain 26.4, and for sands passing a 200-mesh screen and from which all slime was elutriated, 28.1 per cent. of water. The volume relationship is also shown for both fine sands and slime in Fig. 12. The former shows 47.7 of water and 52.3 per cent. of solid; the latter, 60 of water and 40 of solid. These figures are of interest in that they show the comparatively large volume-proportion of water in the slime-cakes.

The comparative results between sand and slime show that the percentage of water is no indication of the permeability of a porous material.

In Fig. 12, *H* shows the results for the Virginia City slime; *I*, for the type slime built up under 21.7-in. vacuum (36.2 per cent. of moisture and 1.662 sp. gr.); *J* shows the same

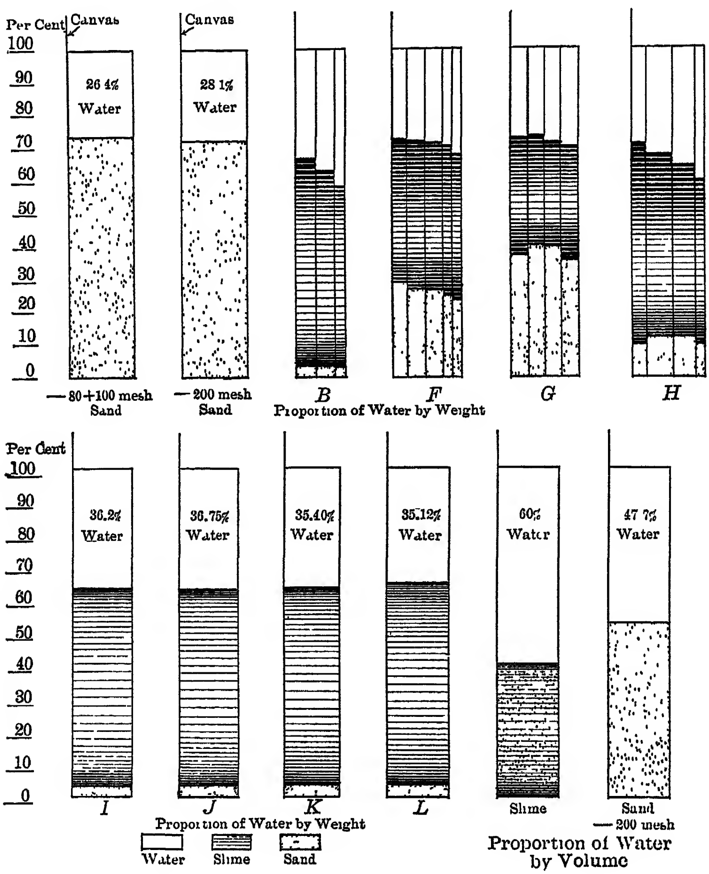

FIG. 12.—SECTIONS OF SLIME-CAKES.

slime built up under 11.35-in. vacuum (36.75 per cent. of moisture and 1.65 sp. gr.); *K* shows the same slime built up under 17-in. vacuum (35.4 per cent. of moisture and 1.675 sp. gr.); *L* shows the same slime built up under 21.8-in. vacuum (35.12 per cent. of moisture and 1.683 sp. gr.).

The data of Fig. 12 serve also to show the effect of pressure

upon the specific gravity of the slime-cake. To these may be added a slime-cake from the type slime which was built up on a slat-supported filter of No. 12 duck, giving with 21.5-in. vacuum a moisture-content of 33.6 per cent. and specific gravity of 1.71; and a cake built up under 30 lb. of air-pressure, giving 28.1 per cent. of moisture and 1.805 sp. gr. With the exception of *K*, there is an increase in the specific gravity with an increase of pressure. This increase in the density of the cake means a decrease in the permeability, and therefore a decrease in filtering-rate. A slime-cake may be likened to a number of layers of rubber spheres. Pressures great enough to overcome the elasticity of the spheres would have the effect of squeezing them into spheroids and reducing the interstitial space; still greater pressures would cause the spheroids to encroach upon the open spaces until these would be practically closed and the interstitial space become zero. The difficulties involved in using high pressures upon slime carrying a minimum proportion of sand are apparent. The effect of the presence of sand would be the same as if angular grains were mixed with the rubber spheres. Under pressure the angular grains would press against one another and prevent any great degree of pressure coming upon the spheres. There would be a comparatively small decrease in the interstitial space, and therefore little or no reduction in permeability.

An interesting phenomenon was noticed in transferring slime-cakes. The lifting of the cake from the pulp was accompanied by an immediate shrinkage in the thickness of the cake. The reduction in thickness amounted to from 10 to 12 per cent. On submerging the cake in the pulp the original thickness would be approximately resumed. The effect of the shrinkage is to increase the density and decrease the filtering-rate. With sand-slime cakes no noticeable shrinkage was observed until the cake approximated a thickness of 1 in., and in all cases this shrinkage was less than that of the-slime cake. During the building up of a slime-cake there is a progressive densification, somewhat slow and often irregular, which accounts for the erratic results obtained in some cases with the filtering-rate experiments.

The progressive densification of a slime-cake may be shown in another way. Subjecting a thick slime-cake to continued

pressure when immersed in water, and determining the filtering-rate at several successive time-intervals, gives a slow drop in the filtering-rate.

Increase in temperature has a slight effect in increasing the density of the cake, but the experiments on this point were on the whole somewhat inconclusive.

The cracking of a slime-cake takes place under two conditions: when it is removed from the pulp and allowed to remain in the air under full pressure, and when removed from a pulp to water at a temperature 40° or 50° C. higher than the pulp. The latter condition is of little practical importance. The former is overcome by reducing the vacuum-pressure to just sufficient to hold the cake upon the cloth. Too long an exposure even at this pressure will cause a cake to crack. Under a vacuum-pressure of 21 in. a 1-in. cake will break down in from 2 to 10 min. Sand-slime cakes will stand a longer exposure than slime-cakes. The cause of cracking is lateral shrinkage, due to the displacement of the water by air and air-drying.

Note.—My attention was brought to the fact that a slime-cake built up from a thick pulp is more apt to crack on exposure than one from a thin pulp. This can be explained by the fact that such a cake densifies more slowly when immersed in a viscous or thick pulp, and consequently is more sensitive when exposed.

In building up cakes with vertically-suspended filtering-cells there is a tendency for the cake to build up thicker at the lower end. This is due to the increase of filtering-pressure due to increased hydrostatic head, and also to the thickening of the slime-pulp in the lower part of the filter-vat. Agitation will prevent, to a large extent, the building up of thick-ended cakes. Where the proportion of sand is large and the sand grains are coarse, agitation is quite necessary, but should not be too vigorous, as otherwise the building up of a cake would be seriously interfered with by erosion. With fine sands, finer than 200-mesh screen, if a pulp-density of 1.4 or more is maintained, little or no trouble is experienced by the sands settling out. Apparently the pulp, in the experiments, remained quite homogeneous for intervals of longer than one hour. With a greater proportion of water in the pulp moderate agitation is necessary in most cases.

Slime-cakes should be built up with vacuum-pressures as constant as possible, and should be kept completely submerged while the cake is forming. The temperature of the pulp and of the wash-water should be the same. Transfers from pulp to wash should be made as rapidly as possible and under reduced pressures. Filters in which the transfers are quickly made, and with the minimum of exposure of the cake to the air, are more efficient and maintain a higher filtering-rate than those in which long time-intervals are required for the necessary transfers.

Building Up Slime-Cakes.

Three direct factors control the rate of and the total time required for building up the cake: the thickness of the cake, the filtering-rate of the slime, and the proportion of water in the pulp. Temperature, viscosity of the pulp, intensity of the vacuum used, depth of submersion of the cell, agitation, and the physical character of the slime, play indirectly a part in the building up of the cake. The effect of the indirect factors is summed up in the filtering-rate.

Practical experience has placed certain well-defined limits upon the thickness of the slime-cake. For example, the Ridgway filter utilizes a thickness of from 0.125 to 0.375 in.; the Oliver, from 0.25 to 0.5; the Butters and Moore, from 0.75 to 1.75; the Merrill, from 1.75 to 2; the Kelly, from 1 to 3, and the Burt revolving-filter, up to 6 in. With vertically-suspended filters the thick cakes tend to tear apart and drop during the transfers. The thickness of the cake also determines the time required for washing. Thick cakes require relatively a much longer time to wash than thin cakes, on account of the low filtering-rates, and the capacity of a filtering-unit may be very greatly cut down. With slime-cakes the lower limits mentioned above are used; with sand-slime cakes the upper limits may be used.

Other things being equal, the less solution that is required to be drawn through a filter in the building up of the cake the more quickly the cake will be secured, and consequently it is desirable to have the slime-pulp as low in content of water as possible. There is a practical limit to the thickening of the slime-pulp, and this is established by the settling-power of the pulp and the fluidity of the settled pulp. The settled pulp

must be handled in pipes and with centrifugal pumps, and if it is too thick it becomes impossible to do this. A thick pulp is advantageous where much sand is to be held in suspension. In Nevada, with quartzose ores, pulp-ratios of from 3 of

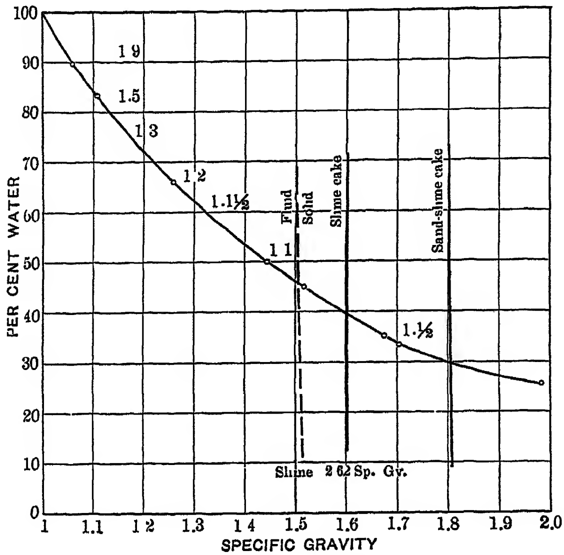

FIG. 13.—RELATION BETWEEN PERCENTAGE OF MOISTURE AND SPECIFIC GRAVITY.

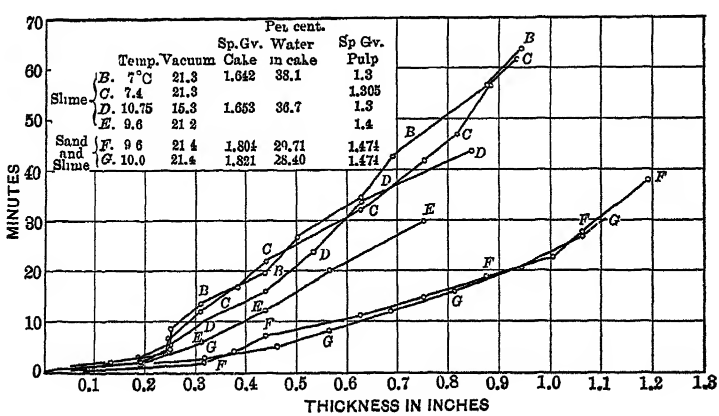

FIG. 14.—RATIO OF BUILDING UP CAKES, NO. 10 CANVAS.

solution to 1 of slime down to 1.5 of solution to 1 of slime are in use. The average pulp in use is 2 of solution to 1 of slime. Fig. 13 shows the relation between percentage of water and

specific gravity of slime-pulp and cakes for a slime of specific gravity 2.62. Similar curves may be worked out for slimes of different specific gravity.

Fig. 14 shows the rate of building up slime-cakes on No. 10 canvas. Curves *B*, *C*, *D*, *E* are for the type slime; curves *F* and *G* are for the type slime with mixtures of sand to the

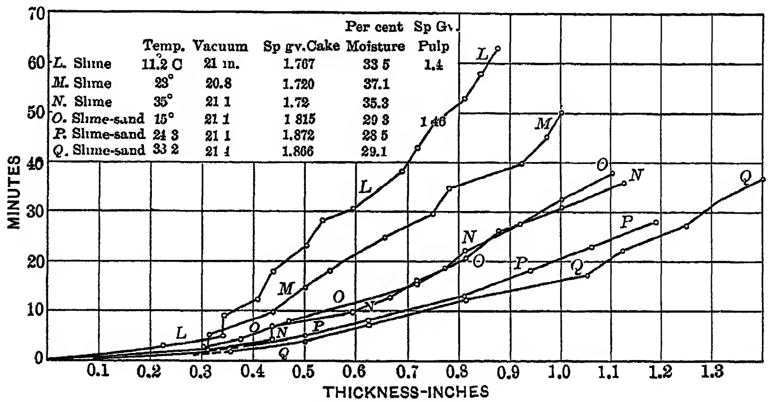

FIG. 15.—RATE OF BUILDING UP CAKES, NO. 12 DUCK.

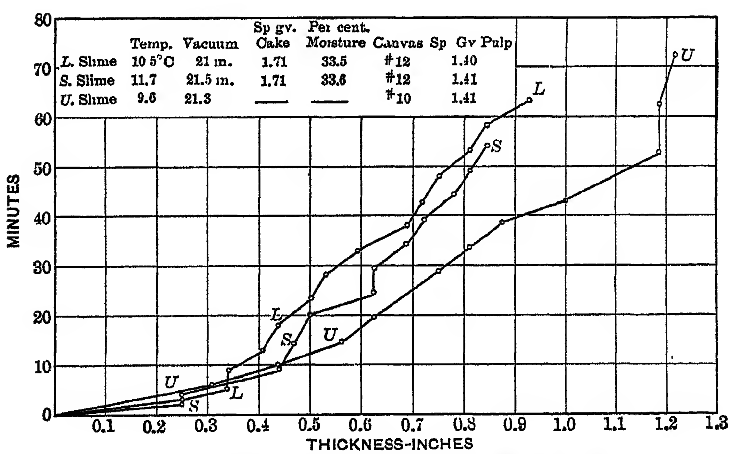

FIG. 16.—RATE OF BUILDING UP SLIME-CAKES.

amount of 37 and 52 per cent., respectively. Curves *B* and *C* show a rapid consolidation of the cake at 0.25 in. thickness, and then a gradual thickening. Curve *D*, built up at a lower pressure, shows a gradual thickening and a somewhat greater rate of thickening than *B* or *C*. Curve *E* shows a greater rate of

thickening on account of the greater density of the pulp. The curves for the sand-slime cakes show a gradual thickening.

Fig. 15 shows the rate of building up slime and sand-slime cakes on No. 12 duck under different temperature-conditions. Curves *L*, *M*, and *N* are for the type slime, and *O*, *P*, and *Q* for a sand slime containing 52 per cent. of sand. All three slime-curves show greater irregularities than the sand slime, and the sudden consolidation of the slime-cakes between 0.3 and 0.45 in. in thickness is characteristic. Increase of temperature increases the rate of building up to a marked extent.

The result of using a more permeable filter is illustrated by comparing Figs. 14 and 15. The significant curves are shown in Fig. 16. Curve *L* is the type slime built up on No. 12 duck

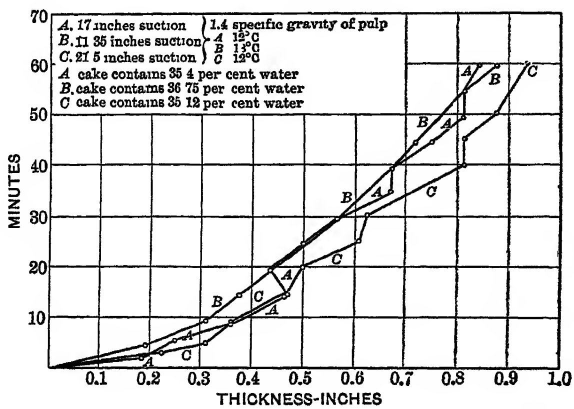

FIG. 17.—BUILDING UP OF CAKE UNDER VARYING PRESSURES.

supported on the grooved board; curve *S* is the type slime built up on No. 12 canvas and diamond-strip support. The more permeable filters show faster rates for the thin cakes and slower for the thick than the less permeable filter with No. 10 duck. The diamond-strip-supported filter gives a faster rate than the grooved board.

Fig. 17 shows the effect of pressure upon the rate of building up. The type slime and No. 10 duck were used in these experiments. Curve *B*, the rate-curve for the lowest pressure, is quite uniform; curve *A* is broken, and shows three consolidations; curve *C* shows corresponding but not such prominent consolidations. The curves, on the whole, indicate that in-

creased pressure, other things being equal, will increase the rate of building up.

Fig. 18 shows the rate of building up a slime-cake from a pulp made up from fire-clay. The very slow rate and the four pronounced consolidations extending over comparatively long time-intervals are of interest.

The rate-curves indicate that as a slime-cake builds up a slow consolidation takes place, and superimposed upon this is

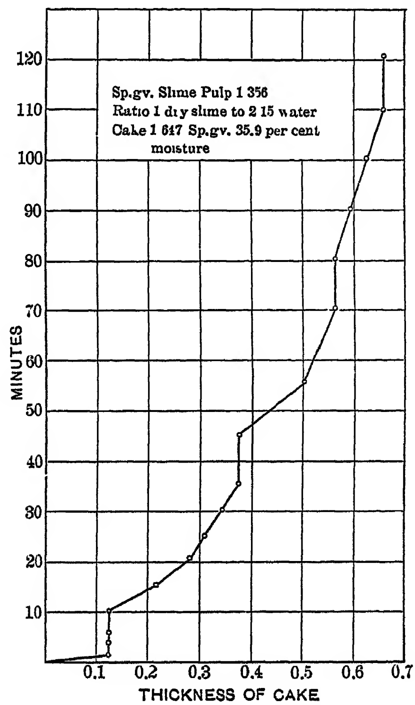

FIG. 18.—BUILDING UP CLAY SLIME.

an irregular and faster rate of consolidation. The irregular consolidation is characteristic of very slimy and clayey slimes, and becomes less so as the proportion of sand is increased. The sand diminishes the elastic nature of the slime-cake.

Washing.

A slime-cake retains from 28 to 38 per cent. of solution; the former for sand-slime and the latter for slime-cakes. Were it not for diffusion, osmosis, and adsorption, simple displacement with a volume of water equal to that retained by the cake would

be sufficient to remove the dissolved salts of gold, silver, and cyanide. As it is, the soluble salts diffuse back into the wash-water, and in time this builds up in gold- and silver-values to such an extent that an appreciable loss results when, as in the Butters filter, the slime-cake is flushed out with the residual wash-water. By the use of two separate wash-solutions this difficulty can be more or less overcome; but it has the objection that an additional pumping of solution and exposure of the slime-cake to the air are necessitated. With the Moore, Oliver, and the pressure-filters generally, no great trouble is experienced from loss of values in the wash-solution, for in each case only that wash-water which is left in the cake after air-displacement takes place is discharged, and this amounts to from 20 to 30 per cent. In practice it is customary to use barren solution or wash-water in amount equal to from one to three times the amount of solution retained by the cake. With very low-grade solutions the former, and with high-grade solutions the latter, would be used. Twice the weight of the solution retained by the cake is usually sufficient to displace the values retained by the solution in the cake.

The thoroughness of washing is determined largely by the expense. When the value of the gold, silver, and cyanide recovered is less than the cost of recovery, washing stops.

It seems to me that with the thin-cake filtering-appliances a smaller proportion of wash-water would be required than for the thick-cake filters, for the reason that the rate of filtration is much higher with the former. Strength of solution, time, and temperature, are the controlling factors in osmosis; and of these, time is perhaps the most important; for the other two would be the same in either case. The shortness of the time required for washing and the relatively high filtering-rate in the case of the Oliver filter would give little opportunity for osmosis to interfere with the washing.

No washing-experiments were made with suction-filters. A few experiments were made with a pressure-filter in washing slime-cakes containing cyanide solutions. With pressures varying from 20 to 30 lb., and with a 0.2-per cent. cyanide solution, a 0.75-in. cake required from 1.3 to 1.4 times the contained water to reduce the cyanide-content to 0.02 per cent.

Complete Cycle of Filtration.

Given the time for making all of the necessary transfers, the time for a complete cycle of operations may be approximated by using the filtering and building-up rate-curves. Two such examples have been worked out in Figs. 19 and 20. The former shows the time required for a complete cycle for the type slime, mixed with 50 per cent. of sand, both for single displacement and double displacement; while the latter shows the time required for the type slime. The 24-hr. capacity per 100 sq. ft. of filtering-surface may be calculated from the diagram, and the specific gravity of the cake and the percentage of moisture retained. Table II. shows the results of such calculations.

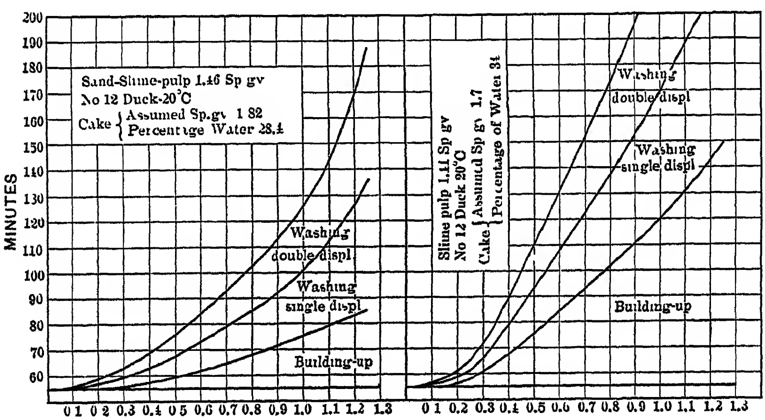

FIG. 19.—CHANGES-TRANSFERS, SAND-SLIME PULP.

FIG. 20.—CHANGES-TRANSFERS, SLIME PULP.

This table is constructed for a Butters filter, and the time for all the transfers is taken as 55 min. These data clearly show the impracticability of using thin cakes for filters of this type. Taking power-costs into account, it is advisable to have as few cycles as possible in filters of the Butters type, and this would be accomplished by building up thick cakes.

It should be noted that the capacities calculated are higher for the slime-cake than obtain in practice, for the reason that a thicker slime-pulp than is ordinarily the custom was used and consequently the time for building up was less.

TABLE II.—*Filtration: Capacity Per* 100 *Sq. Ft. of Filtering-Surface Per* 24 *Hours.*

SAND-SLIME CAKE.

	0 25 In	0 5 In	0 75 In.	1 In.	1.25 In.
Weight of cake, pounds (per cycle),	236	473	710	947	1,183
Weight of dry slime, pounds (per cycle),	170	340	510	681	851
Number of cycles,	24	20.8	17.5	14.2	10.5
Dry slime per single displacement, tons,	2.04	3.54	4.46	4.83	4.46
Number of cycles double displacement,	23 6	18.7	14 6	11.4	7.6
Dry slime, tons,	2.0	3.2	3.7	3.88	3.2

SLIME-CAKE.

	0 25 In	0 5 In	0 75 In.	1 In.	1.25 In.
Weight of cake, pounds (per cycle),	221	442	663	885	1,016
Weight of dry slime, pounds (per cycle),	146	292	438	585	731
Number of cycles single displacement,	22.8	15.1	11	8.4	6.7
Dry slime, tons,	1.6	2.2	2.4	2.45	2.38
Number of cycles double displacement,	21.8	14.1	8 7	6.5	5.1
Dry slime, tons,	1.5	2.04	1.9	1.9	1.8

By using a different horizontal axis in Figs. 19 and 20, the time required for a complete cycle for other suction-filters may be read off and calculations made for capacity.

Data from Slime-Plants.

Data were secured from a number of slime-plants in Nevada through the courtesy of the different managers. Table III. shows the results of a number of physical analyses of slimes obtained from these plants. For purposes of comparison the type slime, the sand-slime mixture, the clay slime, and the filtering-rates are included in the table. The screen-analysis was conducted as follows: A 50-g. sample was taken and mixed thoroughly in a mortar to a thin pulp and then poured into an 800-cc. beaker and sufficient water added to make a volume of 600 cc., or approximately a ratio of 1 of slime to 12 of water. After stirring, beaker No. 1 was allowed to stand 2 min. and the contents poured into a beaker of the same size. The sand left in the bottom was mixed with 600 cc. of water; allowed

TABLE III.—*Physical Analyses and Chemical Composition of Slime.*

PHYSICAL ANALYSES.

	Type.	Sand Slime, 50 Per Cent of Sand.	Fire-Clay Clay Slime.	*A*	*B.*	*C.*	*D.*	*E.*	*F.*	Size.
				Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Millimeter
Less than 2 min. Settlement. + 100 screen	0.10	0.43	22.3	4.5	1.70	2.3	6.2	2.3	6.6	0.17 — 0.22
— 100 + 150	1.30	3.65	14.0	6.9	9.0	12.30	11 3	4.46	6.8	0.11 — 0.17
— 150 + 200	1.20	6.60	4.6	7.3	9.0	13.0	8.35	7.0	10.7	0.05 — 0.11
— 200	25.0	43.80	3.3	39.2	43.70	34.0	38.05	49.54	68.5	0.01 — 0.05
More than 2 min., less than 4 min.,	8.5	6.3	2.0	4.25	4.90	11.0	5.5	3.6	0.7	0.005 — 0.02
More than 4 min., less than 8 min.,	11.6	8.4	2.2	3.6	4.60	6.7	6.2	3.8	0.65	0.005 — 0.01
More than 8 min.,	54.3	30.8	51.6	34.25	27.10	22.7	24.4	29.3	7.05	0.0 — 0.005
Thickness of cake, inches,	1.00	1.00	1.00	1.00	0.75—1.0	1.25—1.75	1.25	1.75	1.5—2	
Filtering-rate {Building, Washing.} Pounds per 100 sq. ft. per minute.	5 lbs.	10	3	11.6 10.0	26 16	8.3	7.25	30		
Sp gr. dry slime,	2.62	2.67	2.57	2.58	2.62	2.62	......	2.684		

CHEMICAL COMPOSITION.

	Type.	Sand Slime.	Fire-Clay Slime.	*A*	*B.*	*C.*	*D.*	*E.*
	Per Cent.		Per Cent.	Per Cent.	Per Cent.	Per Cent.		Per Cent.
Silica,	74	......	68	74	77.2	68	......	64.4
Alumina,	10.9	......	22.9	12.8	13.9	15.8	......	13.5
Ferric oxide,	1.1	......	1.1	1.4	1.9	2.0	......	2.9
Lime,	1.0	......	0.0	0.8	1.0	1.0	......	5.2
Magnesia,	0.6	......	tr.	0.0	0.5	0.0	......	1.2
Aluminum silicate (Al_2O_3 2 SiO_2), calculated,	23.6	......	49.9	27.9	30.3	34.4	......	29.43

to settle 2 min.; contents of beaker No. 2 were then poured into beaker No. 3 and of No. 1 into beaker No. 2; after standing 2 min. contents of No. 3 were poured into No. 4, of No. 2 into No. 3, and of No. 1 into No. 2. No. 1 was filled again and all beakers allowed to stand 2 min. Contents of No. 4 were poured into a 1.5-l. beaker, of No. 3 into No. 4, of No. 2 into No. 3, and of No. 1 into No. 2. No. 1 was left and the steps repeated until all four beakers were empty but for the sands and the reject in two 1.5-l. beakers. The sands were washed into pans and dried and then screen-analyzed through 100-, 150- and 200-mesh screens. The portion of the pulp in the large beakers was stirred and allowed to settle 4 min. and poured off; the sands constituted the 4-min. portion. Stirring and standing 8 min. gave the next to the last portion, and the last portion constituted the remainder. The sands obtained by this method were clean and free from adhering grains and were almost all silica. The size of the grains in the various portions was approximated by measurement with a microscope.

The type and the clay slimes stand out conspicuously. The type slime mixed with 50 per cent. of sand and the mill-slimes compare quite closely. Pulp *F* is a concentrate treated by agitation and pressure-filtration in a Kelly filter. Practically all of the mill-pulp considered as slime passes a 100-mesh screen; 88 per cent. is finer than a No. 150 screen and 79 per cent. finer than a No. 200 screen. The fine portion which takes more than 8 min. to settle gives an average of 27.5 per cent. for the mill-pulps. The chemical investigation was not completed, but the results of partial analyses are sufficiently interesting to include. With two exceptions, the type slime and mill-slime *C*, the calculated percentage of aluminum silicate corresponds approximately with the portion of the pulp taking longer than 8 min. to settle. Microscopical examination of the coarser portions show them to consist almost entirely of silica. The results on the type slime indicate that considerable silica remains with the last portion of the slime. Slime *C* is from the Goldfield Consolidated mill; as is well known, part of the alumina is present as alunite, and consequently the calculation of all of the alumina to aluminum silicate gives a result too high. The results of the chemical analyses indicate

that by combining physical and chemical methods an approximate separation and quantitative determination of crystalline and colloidal material may be effected.

Table IV. gives the data of the slime-plants from which the mill-slimes in Table III. were taken.

TABLE IV.—*Details of Slime-Plants.*

	A.	*B.*	*C.*	*D.*	*E*	*F*	*G.*
Type of filter, . . .	Butters.	Butters.	Butters.	Butters.	Merrill.	Kelly	Butters.
Number of units, .	2	2	2	2	3	1	1
Number of leaves per unit, ...	60	95	168	72	64	10	100
Number of leaves, .	120	190	336	100	192	..	100
Area of leaf, sq. ft., .	100	91	100	92.6	41 4	..	100
Total filtering-area, .	12,000	17,290	33,600	9,262	7,980	360	10,000
Tons slime per 24 hr.,	175	250	1,000	150	216	100	150
Tons slime per 100 sq. ft. per 24 hr., . .	1 4	1.44	2.97	1.62	2.72	27.7	1.5
Slime-pulp consistency, water : slime,	3.1	3:1	1.5 1	2.1	2:1	1'.1	2.1
Filter-support, . .	slats	slats	matting	slats	plate	wire netting	slats
Thickness of cake, in ,	1	0 75–1	1.25–1.75	1.25	1.75	1.5–2	1 5
Moisture in cake, p.c.,	38	35	35	33	.....	12	29
Time forming, hr ,	1	0 5	1–1.33	1	0.66	0.03	2
Time washing, hr , .	1	1.0	1.41–1.66	1.25	0.41	0.20	2
Time transfers, hr ,	0 75	0 83	0.59–0.51	0 5	3.01	....	1.83
Total time-cycle, hr ,	2.75	2.33	3–3.5	2.75	4.08	0.5–0.75	5.83
Filtering-rate per 100 sq. ft. per min., .	11.6	26	8 3	6 83	30	..	......
Filtering-rate per min. for wash, . .	10	16.3	......	......	......	......	....
Tons of solution and wash per 24 hr., .	848	1,356	2,000	350–400	328 340	*a*75	484
Canvas used, oz., .	12	12	12	12	18	12	10
Life of canvas, months,	8	6	41 est.	18	12	0.5–1.5	
Frequency of acid-wash, days, . . .	20	21	30	21	60	15	30
Approximate cost per ton slime, ...	\$0.18	\$0.27	\$0.075	\$0.25	\$0.238	\$0.35	\$0.119

a 8–12 hrs.

Conclusions.

Certain practical conclusions may be drawn from the experimental work. These have been in part stated, but may not be out of place here, together with certain general conclusions which are not so directly shown by the experimental work. They are:

1. The proportion of clayey material in ores which are to be subjected to "all-sliming" and filtration should be maintained at a minimum.

2. The slime-pulp should be as free as possible from sands coarser than a No. 150 screen, and as large a proportion of the

pulp as possible should consist of material passing a No. 200 screen.

3. The slime-pulp before filtration should be settled to as thick a consistency as possible consistent with ready handling by pumps and in pipes.

4. The temperature of the slime-pulp should be maintained between 20° and 30° C. or higher.

5. The temperature of the wash-water and the pulp should be the same.

6. Vacuum-pressures should be varied until the proper intensity for the given slime is obtained.

7. Where very clayey slime is to be filtered, as much fine sand (limited as stated above) should be crowded into the pulp as it will carry without undue settling and clogging.

8. No. 10 canvas supported by slats gives the best all-round service for the thick cake, and No. 12 canvas on wire netting answers the requirements for the thin-cake filtering-machines.

9. With slimes containing a large proportion of colloid or clayey material pressures greater than those obtainable with vacuum apparatus are of questionable advantage.

10. With slimes containing a large proportion of clayey material the vacuum-filters should be used.

11. With slimes containing a small proportion of clayey material and much fine sand both vacuum-filters and pressure-filters could be used with perhaps equally good results.

12. With slimes containing much coarse and fine sand the chamber-filters with air-agitation and high pressures would perhaps give the best results.

13. Of the vacuum-filters, the thin-cake continuous filters are a decided improvement over the thick-cake filters.

Acknowledgments.

I am especially indebted to many of the students of the Mackay School of Mines, to Jay A. Carpenter, to W. S. Palmer, and to many of the superintendents and managers of milling-plants in Nevada for assistance and data. In closing this paper I wish to express regret for my inability, on account of the pressure of other duties, to carry out more completely closely-related lines suggested by the experimental work.

The Cyanide-Plant at the Treadwell Mines, Alaska.

BY W. P. LASS, TREADWELL, ALASKA.*

(San Francisco Meeting, October, 1911.)

THE purpose of this article is not only to describe the plant and method of cyaniding the Treadwell concentrates, but to present some of the results of the experimental work obtained in the past three years for the Alaska-Treadwell Gold Mining Co., at Douglas Island, Alaska, under the direction of F. W. Bradley, Consulting Engineer, and Robert A. Kinzie, General Superintendent, of the affiliated companies.

At the time the experimental work was undertaken the concentrates were being shipped to the smeltery at Tacoma, Wash., and the cost for treatment of 3-oz. (gold) concentrates was $11.95 per ton, divided as follows:

Smelting-charges,	$4.00
Loading, freight, insurance, etc.,	2.89
Interest due to time lost in transit and in settlement,	0.05
Loss due to settlement for 95 per cent. of the gold at $20 per ounce,	5.01
Total,	$11.95

From the experimental work described later, it was estimated that 96 per cent. extraction could be made by treatment on the ground, and that the cost, when treating 80 tons per day, would be $3.25 per ton, divided as follows:

	Per Day.	Per Ton.
Labor,	$66.16	$0.827
Chemicals,	76.60	0.960
Power and steam-heat,	67.60	0.845
Marketing-, refining- and other charges,	49.36	0.617
Totals,	$259.72	$3.250

Adding to this total the 4 per cent. treatment-loss, which on 3-oz. concentrates amounts to $2.48, gives a total cost of $5.73 per ton. Comparing this with $11.95, the cost when shipping to the smelter, leaves a net gain of $6.22 per ton by the local

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treatment. In addition to this saving, the cyanide-tailings would have an economic value due to the sulphur- and iron-content, as well as the value of the residual gold after oxidation.

I. Laboratory-Work.

1. *Character of the Concentrates.*—The concentrates, amounting to 1.8 per cent. of the original ore, contain: Fe, 40; S, 40; SiO_2, 11 per cent., and carry from 2.5 to 4 oz. of gold and 0.75 oz. of silver per ton. The gold- and silver-values amount to about 37 per cent. of the values contained in the original ore from the mine. The figures in Table I. are assays and averages of sizing-tests on concentrates from the various mills.

Table I.—*Assay Sizing-Tests of Treadwell Gold- and Silver-Ores.*

Size of Material.	Weight.	Assay-Value Per Ton.	Value.	Value in One Ton of Original.
	Per Cent.		Per Cent.	
On 20-mesh screen.	0.44	$70.35	0.48	$0.31
Through 20, on 40......	8.23	203.96	26.05	16.83
Through 40, on 60	10.96	143.89	24.39	15.76
Through 60, on 80	12.49	94 88	18.34	11.85
Through 80, on 100.......	10.38	60.85	9.78	6.32
Through 100, on 120	13.37	39.27	8.14	5.25
Through 120, on 150.... ..	7.69	26.61	3.17	2.05
Through 150..................	36.46	17.10	9.65	6.23
	100.00		100.00	$64.60

(In this paper, all figures, unless otherwise stated, are based on the dry ton of 2,000 lb., with gold at $20.67 per oz. Silver-value is not included. Screen-mesh is expressed in openings per linear inch.)

On account of the decrepitation of the pyritic crystals during the process of drying, as well as the tendency of the particles to adhere to one another, all sizing-tests were made in water without previous drying of the sample. Results show that the values vary directly with the degree of comminution. It being understood that the concentrates are derived from pulp after amalgamation at the mills, it seemed evident that the gold was present as metallics incased within the pyrite. Work done in the laboratory previous to the year 1909 confirmed this view, and indicated that a satisfactory extraction could be obtained by regrinding, followed by amalgamation and cyanidation.

2. *Preliminary Tests.*—For the preliminary tests ordinary quart-size glass jars were used, and agitated by placing them on the distributing-boxes of the Frue vanners. In each case an excess of lime and a small amount of lead acetate were added to the solution. Sizing and assaying of the residues showed the gold to have been removed from the finely-ground particles, while the large percentage of value remained in the coarse particles.

The next step was with 50-lb. composite samples from all the mills. A clean-up barrel was fitted with iron balls and used to grind the concentrates to a 200-mesh product, which was passed over a 2- by 4-ft. amalgamated copper plate, the pulp collected and cyanided in small agitation-vats, built on the plan of "Brown" or "Pachuca" tanks. These were 14 in. in diameter and 4 ft. high, with a 1.25-in. pipe suspended through the center. At the apex of the cone a needle-valve regulated the supply of air.

The 50-lb. samples were treated in these small tanks, the results given in Table II. being a fair average from one of these tests.

TABLE II.—*Results Obtained from Treatment of 50-Lb. Composite Samples from Treadwell Mills.*

Assay-value of original concentrates, ...	$77.40
Amalgamation-extraction based upon head- and tail-assays, per cent., ...	74.16
Proportion of ground product passing 200-mesh screen, per cent.,	98.00
Assay-value of cyanide heads, ...	$20.00
Assay-value of cyanide tails, ...	$2.40
Cyanide-extraction based upon head- and tail-assays, per cent., .	88.00
Cyanide-extraction based upon solution-assays, per cent., . .	90.00
Total extraction by amalgamation and cyanide, per cent., . .	96.89
Time of cyanide-treatment, hours, ...	12
Strength of cyanide solution (1 lb. per ton), per cent., . .	0.05
Cyanide-consumption per ton of concentrates, pounds, . .	2.6
Lime-consumption per ton of concentrates, pounds, . . .	14.0

The tests in Table II. show that 75 per cent. of the gold could be obtained by fine grinding and amalgamating, or 96 per cent. by fine grinding and amalgamating followed by cyaniding.

II. Experimental Plant.

Having proved that a satisfactory extraction could be obtained, the next step was to determine the most economical method of handling the material. For this purpose, an addition was built to one of the mills, in which was installed an Abbé 4- by 12-ft. tube-mill, with the necessary plates for amalgamation. The tube-mill ground 0.5 ton of concentrates per hour to pass a 200-mesh screen, or 1 ton per hour, 95 per cent. of which would pass a 200-mesh screen. With a cleaner separation of the coarse return-product, the grinding-capacity could have been increased. Various forms of classifiers were tried, the Dorr "drag" classifier proving the most satisfactory, not only making a good separation between the sands and fines, but acting as a feeder to the tube-mills. In later practice, with a duplex Dorr classifier treating 125 tons daily of concentrates discharged from a larger tube-mill, the following results were obtained:

	Screen Mesh. On 100. Per Cent.	On 200. Per Cent.	Through 200. Per Cent.
Feed to classifier, . .	10.1	26.4	63.5
Coarse discharge, . .	51.3	44.0	4.7
Fine overflow, . . .	1.1	29.7	69.2

As ordinarily used, the water is much in excess of the ore, so that the fines are carried over by the rising current from the rakes; but in operating the Dorr to its fullest capacity on concentrates, it is necessary to reduce the volume of water used, and depend upon the greater specific gravity of the pulp holding the fines in suspension until carried over with the fine product.

Callow cones arranged with suspended diaphragms were used for de-watering the sands previous to cyaniding. When delivering a clear overflow, one standard 8-ft. cone was found to have an hourly capacity of 1 ton of concentrates with 15 tons of lime-water, making a spigot-product with less than 35 per cent. of moisture.

Grinding in an alkali solution equivalent to 2 lb. of lime per ton kept the amalgamation-plates in a clean, bright condition, and materially aided in the settlement of slimes. Without lime the pulp discharged from the tube-mill possessed a latent acidity equivalent to 6 lb. of lime per ton of concentrates,

which made plate-amalgamation almost impossible on account of a black surface-deposit completely coating the plates within 10 min. after being dressed.

Sea-water as a substitute for lime-water was tried, and although it gave better amalgamation-results than fresh water, it was not as satisfactory as the lime solution. The plates became coated with slime and the solution remained turbid in the tanks.

By fine grinding and amalgamating in 15-ton lots, an extraction of from 75 to 80 per cent. was obtained, the extraction varying directly with the fineness of grinding. On the original ore this amounts to an extraction of 84 per cent. by amalgamation.

To obtain the best results by amalgamation, mercury was fed into the tube-mill with the concentrates. After having completed the amalgamation-tests, during which time 7,050 oz. of amalgam was recovered, the mill was emptied of its pebbles and the inside thoroughly cleaned, in order to determine the amount of mercury or amalgam that remained. No free mercury and only 3 per cent. of the total amalgam was recovered from the tube.

Upon again feeding the concentrates to the tube-mill without either cyanide or mercury, a concentration took place inside the mill, as shown by the daily sampling of the feed and discharge of the mill, Table III.

TABLE III.—*Results Obtained by Treatment of Concentrates in the Tube-Mill.*

	Original Feed from Bins.	Tube-Feed (Includes Coarse Return Product).	Pulp as Discharged from Tube-Mill.	Slime Finer than 200-Mesh.
First 6 hr. grinding........	$48.00	$95.00	$88.00	$18.00
Second 6 hr. grinding......	48.00	113.00	103.00	16.90
Third 6 hr. grinding.........	48.00	131.00	120.00	19.20

Cyanide was then introduced into the grinding-solution and samples assayed as follows:

	Strength of Cyanide in Grinding-Solution	Original Feed from Bins.	Tube-Feed (Includes Coarse Return Product).	Pulp as Discharged from Tube-Mill.	Slime Finer than 200-Mesh.
	Per Cent.				
First 6 hr. grinding........	0.05	$48.00	$96 00	$80.00	$14.60
Second 6 hr. grinding.....	0.05	48.00	67.00	62 00	11.60
Third 6 hr. grinding.......	0.046	48.00	68.40	59.20	12.00

The method of grinding proving successful, the next step was to test the cyanide process on a larger scale. For this purpose a Brown or Pachuca agitation-tank, 10 ft. in diameter and 22 ft. high, with 60° conical bottom, was erected beside the tube-mill, together with four small redwood tanks. A Merrill precipitation-press was later purchased and a few filter-leaves placed on the suction of the gold-pump for clarifying the solutions. This completed the necessary equipment for cyaniding the tube-mill product in 15-ton lots. The gold-values were removed from the pulp by successive washes and decantations.

TABLE IV.—*Results of Zinc-Dust Precipitation, Obtained in Experimental Plant.*

Cyanide Per Ton of Solution.	Lime Per Ton of Solution	Gold Before Precipitation	Gold After Precipitation.	Gold Precipitated.
Pounds.	Pounds.			Per Cent.
0.44	0.42	$13.60	$12.40	8 82
0.80	0.46	13.00	13 00	0.00
0.92	0.95	6.60	2.20	66.67
1.53	1.07	1.20	0.05	98.81
0.40	1.30	12.10	2.50	79.26
0.40	1.30	4.50	1.60	65.25
1.24	1.35	17 00	0 10	99.41
1.76	1.35	14.60	0 05	99.66
0.80	1.41	3.40	0.20	93.23
1.93	1.47	7.60	0.05	99.34
1.00	1.85	12.80	0.05	99.61
0.98	1.91	5 20	2.80	46.15
2 44	2.18	4.80	0.10	97.92
2.76	2.35	12.60	0.05	99.53

The figures of zinc-dust precipitation presented in Table IV. show the non-precipitation of the values when the lime-content of the solutions fell much below 1 lb. per ton. With solutions high in lime an excess of cyanide was added to keep the filter-cloths clear. In each case an excess of zinc-dust was added.

The flow-sheet, Fig. 1, shows diagrammatically the method used for these experiments, with the exception that the filter-box shown was later superseded by a Kelly filter-press (type 1 B) of 50 tons daily capacity, which did away with the numerous washes and decantations previously required.

The cycle of operations of the Kelly press and the time of working, when forming a 1-in. cake of about 4 tons of concentrates (dry weight), are as follows:

Operation of the Kelly Press.

	Time. Minutes.
Filling press,	3
Forming cake,	2
Returning excess pulp,	2
Washing,	12
Returning excess wash,	2
Drying,	8
Opening, discharging, and closing,	15
Total time of one cycle,	44

Moisture in pulp fed to press,	35 per cent.
Moisture in tailings cake discharged,	8 to 10 per cent.
Pressure of forming cake,	30 lb per sq. in.
Amount of wash-water used per ton of concentrates,	0.5 ton.

The first 25 test-runs made in the experimental plant are summarized in Table V.

TABLE V.—*Results of First Twenty-Five Experimental Test-Runs on 700-Foot Mill-Concentrates.*

No. of Test.	Tons Treated.	[a] Assay Per Ton.		Treatment During Grinding.	Extracted During Grinding.	Ground Product Through 200-Mesh.	Ratio Ore to Solution.	[a] Assay Per Ton.		Extracted by Cyanide in Pachuca.	Cyanide Solution Used.	Lead Acetate Per Ton Concentrates.	Time of Agitation.	Cyanide-Loss Per Ton Concentrates	Changes of Solution.	Total Extraction.
		Original Concentrates	Ground Product.		Per Cent.	Per Cent		Agitation-Heads.	Cyanide-Tails.	Per Cent.	Per Cent.	Pounds	Hours.	Pounds.	No	Per Cent.
1	20	$42.16	$19.63	Amalgamation........	54.0	94.0	1 to 1.6	$19.63	$4.00	79.0	0.35	...	48	6.70	2	90.0
2	15	54.80	9.00	Amalgamation	83.0	99.8	1 to 2.2	9.00	.95	89.0	0.15	1.5	18	4.00	3	98.3
3	18	42.00	10.40	Amalgamation ...	75.2	99.6	1 to 2	10.40	1.40	86.4	0.125	1.14	16	2.97	3	96.6
4	16	47.20	10.00	Amalgamation . . .	78.8	99.8	1 to 2	10.00	1.20	88.0	0.1	1.0	16	1.92	3	97.8
5	10	32.00	8.40	Amalgamation	73.7	97.0	1 to 3	8.40	.80	90.4	0.14	1.0	24	2.00	2	97.5
6	10	56.80	9.60	Amalgamation...	83.1	97.0	1 to 3	9.60	1.80	81.2	0.07	. . .	10	2.30	3	96.8
8	8	50.00	8.80	Amalgamation	82.4	98.0	1 to 3	8.80	2.30	73.8	0.038	1.0	18	1.44	2	95.4
9	9	30.80	9.00	Amalgamation ..	70.7	97.0	1 to 2	9.00	1.60	71.7	0.1	...	12	1.08	2	94.8
10	15	52.00	14.60	Amalgamation.. ...	71.9	90.0	1 to 2.1	14.60	2.00	87.6	0.15	1.0	24	0.90	2	96.1
11	15	37.20	13.40	Amalgamation 0.002 per cent. cyanide.	64.0	88.0	1 to 2	13.40	2.40	82.0	0.15	1.0	24	0.96		93.6
12	15	40.80	9.60	Amalgamation 0.05 per cent. cyanide.	76.5	98.0	1 to 2	9.60	2.40	75.0	0.15	..	24	4.46		94.0
13	15	46.00	9.40	Amalgamation 0.025 per cent. cyanide	79.6	100.0	1 to 2	9.40	1.70	82.0	0.15	0.67	24	2.20		96.3
14	15	36.80	9.80	Amalgamation 0.05 per cent. cyanide..	73.4	100.0	1 to 2	9.80	1.60	83.6	0.15	......	24	3.50	2	95.0
15	15	36.00	5.00	0.07 per cent. cyanide	86.1	100.0	1 to 2	5.00	1.12	77.6	0.15	1.0	24	4.40	2	96.9
16	7	48.00	18.00	Concentration	62.4	100.0	1 to 4	18.00	.90	95.0	0.15		18	1.60	2	98.5
17	7	48.00	16.90	Concentration	64.7	100.0	1 to 4	16.90	1.10	93.5	0.15	....	18	1.36	3	97.6
18	9	48.00	19.20	Concentration	60.0	99.0	1 to 3	19.20	1.80	90.6	0.075		18	2.28	3	96.2
19	12	48.00	13.20	0.05 per cent cyanide.	72.4	99.0	1 to 2	13.20	2.60	80.3	0.1	..	12	5.10	2	94.6
20	10	48.00	12.00	0.046 per cent. cyanide	75.0	99.0	1 to 2.2	12.00	2.60	78.3	0.033	..	18	4.80	3	94.5
21	10	32.00	14.00	0.018 per cent. cyanide	56.2	99.0	1 to 2.3	14.00	2.80	80.0	0.036	..	36	3.70	4	91.2
22	18	28.00	22.40	0.024 per cent cyanide	20.0	97.0	1 to 2	22.40	2.40	89.2	0.025	. ...	24	2.90	3	91.4
23	16	33.00	12.20	0.032 per cent. cyanide	63.0	99.0	1 to 2	12.20	.90	92.6	0.125	.	24	4.70	3	97.2
24	16	55.00	15.35	0.028 per cent. cyanide	72.1	99.0	1 to 2	15.35	1.60	89.5	0.12	....	16	5.20	3	97.8
25	15	46.00	17.60	0.042 per cent. cyanide	61.8	99.0	1 to 2	17.60	2.08	88.2	0.12		16	5.00	2	95.4
26	15	55.00	8.00	0.066 per cent. cyanide	85.4	99.0	1 to 2	8.00	2.90	63.7	0.12		16	4.30	2	94.8

[a] All values are in gold at $20 per ounce. Tests 2, 6, 8, 9, 10, 16 and 17 given preliminary alkali agitation
All solutions contained an excess of lime.

The actual net value of the bullion recovered by amalgamation was 3.7 per cent. in excess of the theoretical extraction figured from head- and tail-assays. The actual value of the precipitate recovered was 5 per cent. in excess of the theoretical extraction.

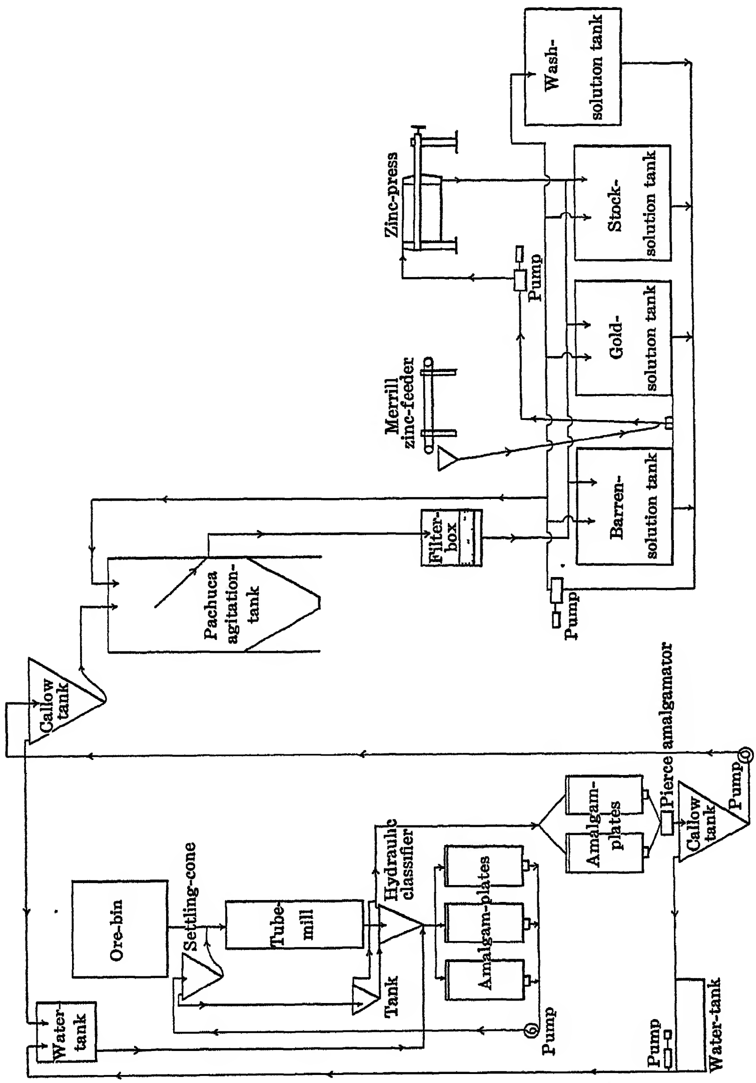


Fig. 1.—Flow-Sheet of Experimental Cyanide-Plant.

The results of the tests showed that 75 per cent. of the gold could be recovered by grinding and amalgamating, or 96 per cent. by the combined method of amalgamating and cyaniding.

Results also showed that during the process of grinding in 1.5-lb. (0.075 per cent.) cyanide solution, a similar extraction could be obtained without amalgamation. Thus a satisfactory extraction was obtained either by amalgamating and cyaniding or by cyaniding direct.

A preliminary agitation with an alkali solution was found to shorten the time of cyanide treatment and save 25 per cent. in the cyanide-consumption.

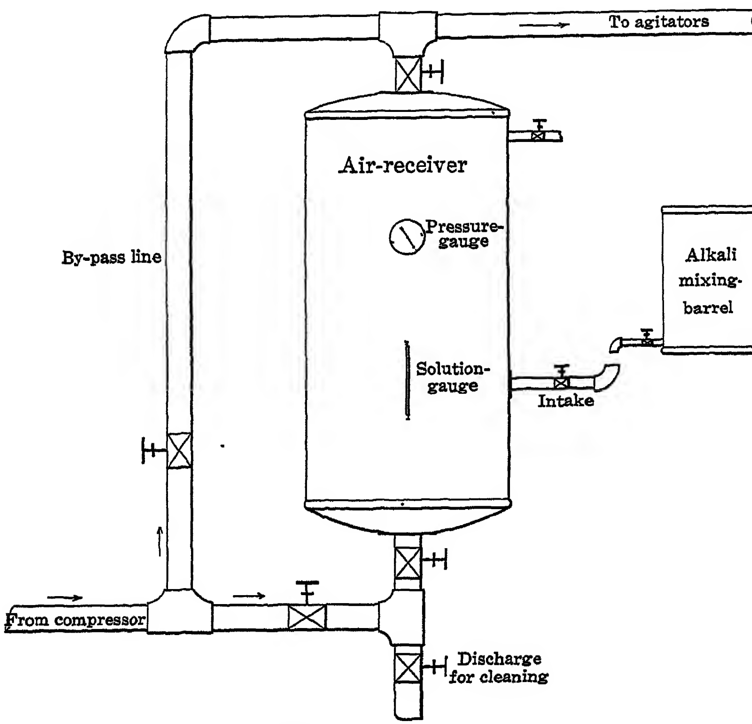


Fig. 2.—Method of Purifying Air.

Passing the air used for agitation through a receiver filled with a solution of caustic soda or milk of lime also decreased the cyanide-consumption, presumably by the removal of oil and carbonic oxides from the air. The pipe-connections illustrating the method of adding the alkali solution are shown in Fig. 2.

When grinding in cyanide solution stronger than 1 lb. per ton (0.05 per cent.), followed by amalgamation, it was difficult

to keep the plates bright, due to a dull white surface-deposit, which if allowed to remain turned to a dull gray. A Muntz-metal plate was substituted for a copper plate, but as all the plates were silver coated no variation in the result was noted.

The results obtained from this extended period of investigation, lasting over two years and at a cost of $27,794, justified the building of a plant of 100 tons daily capacity. This cost was largely offset by the ability of the final plant to treat the concentrates without the usual alterations necessary in starting a new mill. It also formed the nucleus of the final mill-crew.

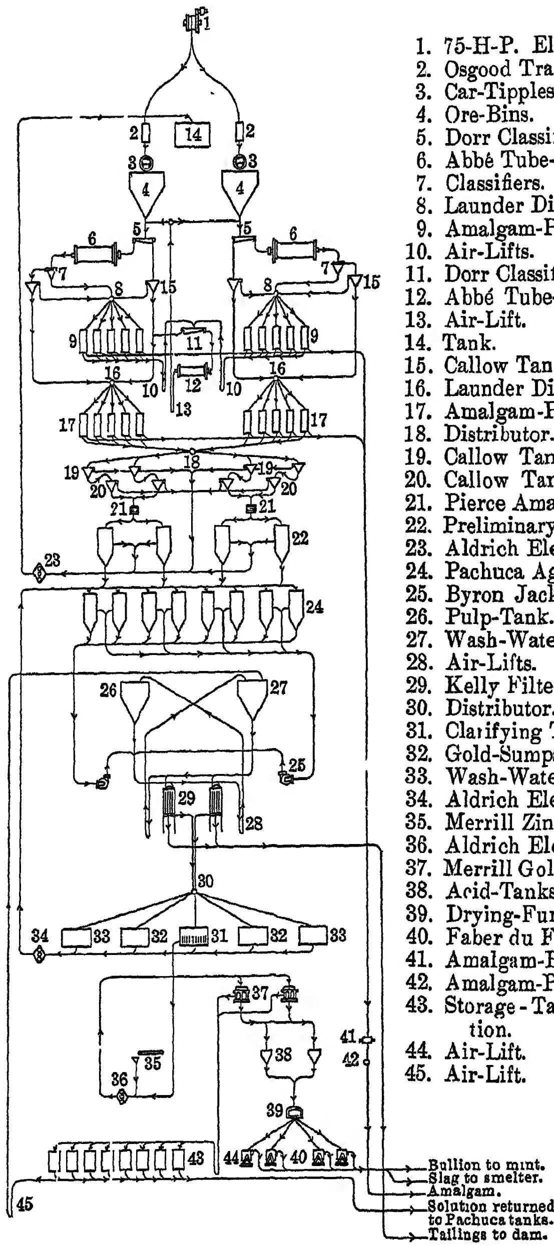
As the abandonment of amalgamation of high free-gold values in favor of direct cyaniding seemed a somewhat radical change, the new mill was planned for operating either way, and ultimately nearly 5,000 tons were treated by each method before deciding to cast out the time-honored amalgam-plate. All of the equipment purchased for the experimental work was used in the permanent plant, which was completed in September, 1910.

III. The 100-Ton Cyanide-Plant.

The cyanide-plant consists of three main buildings located on a hill-side 200 ft. above the stamp-mills. The upper building contains the grinding-and-amalgamating plant, with a lower floor for solution-storage tanks. The lower contains the cyanide equipment proper, while the refinery is in a concrete building at one side, as shown in Fig. 3.

The five mills on the island contain a total of 900 stamps, and crush approximately 5,000 tons daily. The crushed ore after amalgamation is concentrated on 360 Frue vanners, yielding an average of 90 tons of concentrates daily, of from 2.5 to 4 oz. of gold per ton.

From the vanner-boxes the concentrates are shoveled into specially-constructed flat-bottomed steel cars. These cars, each holding 2 tons of concentrates, are made up into trains at the mills, and brought by locomotives to the foot of the incline below the cyanide-plant. This incline is 900 ft. long with 14° rise. A Union Iron Works geared hoist, driven by a 75-h-p. electric motor, brings the train to a switch above the upper building. Beginning with this switch, the entire plant is in duplicate throughout. A flow-sheet of the operations is shown in Fig. 4.



1. 75-H-P. Electric Hoist.
2. Osgood Track-Scales
3. Car-Tipples.
4. Ore-Bins.
5. Dorr Classifiers.
6. Abbé Tube-Mills.
7. Classifiers.
8. Launder Distributors.
9. Amalgam-Plates for coarse pulp.
10. Air-Lifts.
11. Dorr Classifier.
12. Abbé Tube-Mill
13. Air-Lift.
14. Tank.
15. Callow Tanks.
16. Launder Distributors.
17. Amalgam-Plates for fine pulp.
18. Distributor.
19. Callow Tanks.
20. Callow Tanks.
21. Pierce Amalgamators.
22. Preliminary Agitation-Tanks.
23. Aldrich Electric Triplex Pump.
24. Pachuca Agitation-Tanks.
25. Byron Jackson 4-in. Cent. Pumps.
26. Pulp-Tank.
27. Wash-Water Tank.
28. Air-Lifts.
29. Kelly Filter-Presses.
30. Distributor.
31. Clarifying Tank.
32. Gold-Sumps.
33. Wash-Water Sumps.
34. Aldrich Electric Triplex Pump.
35. Merrill Zinc-Feeder.
36. Aldrich Electric Triplex Pump.
37. Merrill Gold-Precipitation Presses.
38. Acid-Tanks.
39. Drying-Furnace for Precipitate.
40. Faber du Faur Furnaces.
41. Amalgam-Barrel
42. Amalgam-Press.
43. Storage-Tanks for barren solution.
44. Air-Lift.
45. Air-Lift.

FIG. 4.—FLOW-SHEET OF 100-TON CYANIDE-PLANT.

Leaving the switch by gravity, the cars are weighed, sampled, and run into revolving tipples. Upon releasing the brake the tipple revolves, turning the car bottom up and dropping the load from the car. The change in the center of gravity then causes the tipple to right itself, and the empty car is weighed and returned to the main switch.

Most of the water is removed from the concentrates while in the vanner-boxes by the aid of a bumper, which is simply a large air-piston machine mounted on a truck and moved from box to box. This bumping causes the concentrates to readjust themselves and pack in the bottom of the box, while the water is run off, leaving about 12 per cent. of moisture in the concentrates. It is considerably easier to shovel the concentrates into the cars after the bumping.

The concentrates are sampled while in the car by means of a long ship-auger. With the ordinary long spoon it was impossible to obtain satisfactory checks in the samples, as the concentrates are usually covered with water. Unslaked lime is added to each of the empty cars as it leaves the tipple in order to reach the concentrates at the earliest possible stage. It also forms a line of cleavage, causing the concentrates to dump clean from the bottom.

From the cars the concentrates fall into 100-ton steel storage-bins, 15 ft. in diameter, with 55° conical bottoms. The concentrates in the bins are kept covered with water, which effectually prevents oxidation of the "sulphurets" while lying in the bins. From this point until the cyanide treatment begins the concentrate is in strong lime solution at all times.

At the apex of the conical bottom of each bin tight-fitting gates control the outflow of concentrates, which is at once sluiced directly into Dorr classifiers, Fig. 5. The sluicing medium is the coarse return-product referred to later. There are three Dorr classifiers driven by one 7.5-h-p. electric motor, one feeding into each tube-mill and making 24 strokes per minute. This rate of speed, causing greater agitation, was found necessary to separate the large bulk of the fine from the coarse.

The coarse product of the classifiers falls into the spiral feeders of the tube-mills. These mills are of the Abbé type, 5 by 22 ft., lap-welded, trunnion bearings, with corrugated sec-

tional liners; 3-in. Danish flint pebbles are used for the grinding.

Two 75-h-p. motors on three-phase circuit at 2,200 volts are belted to an overhead central line-shaft, which in turn is belted to the pinion-shaft of the tube-mills. The tubes are driven from the discharge ends and make 27 rev. per min. The mills are controlled by friction-clutch pulleys on the central line-shaft.

For the period from May 15 to July 15, 1911, one tube-mill ground at the rate of 88.75 tons of concentrates per 24 hr. actual running-time, the power-consumption averaging 64 h-p. By replacing each 75-h-p. motor with a 100-h-p. motor, and substituting leather for canvas belts on the main drive, the power-consumption was reduced to an average of 59 h-p. for the same tube-duty. This was with the tube just half filled with pebbles, the normal running-load. By increasing the pebble-load to 6 in. above the center of the tube, the power-consumption rises to 75 h-p., and both the quantity of tube-feed and the fineness of the product discharged are increased.

The following is an average screen-analysis of the feed and discharge of one 5- by 22-ft. mill, when grinding an original feed of 88.75 tons per 24 hours:

	On 100-Mesh. Per Cent.	On 200-Mesh. Per Cent.	Through 200-Mesh. Per Cent
Feed,	48.7	41.5	9.8
Discharge,	10.1	26.4	63.5

The pulp contained 38.5 per cent. of moisture.

When the concentrates are amalgamated previous to cyaniding, the product discharged from the tubes is distributed over 10 copper amalgamating-plates, each 4 ft. 8 in. wide by 10 ft. long, plated with 2 oz. of silver per square foot.

The pulp flows from the plates into launders built into the floor. No traps are used, as they are quickly clogged by the metallic iron which accumulates in the concentrates from the wear of the various machines used in the processes of mining and milling.

This iron, if allowed to accumulate in the coarse return-product, will amount to as much as 15 per cent. of the total. Experiments are now being carried on with a magnetic device for removing the iron from the pulp.

From a sump in the launder an air-lift elevates the pulp to a *spitzlutte*, from which the coarse material is continuously drawn into a Dorr classifier, the coarse from which feeds a 4- by 12-ft. Abbé tube-mill, similar to the larger ones described above. The discharge from this mill joins the overflow from the *spitzlutte*, and is elevated by air-lifts to two settling-cones, so situated that the spigot-discharge from them becomes the sluicing medium for the original feed referred to above.

Two points will be observed here: (1) that the Dorr classifiers are at present doing all the classifying for the mill; and (2) that the concentrates are carried around in a closed circuit from which there is no escape until the particles have become fine enough to join the overflow from the back of the Dorr classifiers.

The Dorr overflow, which is the product cyanided, is more than 98 per cent. through 200-mesh. The remaining 2 per cent. is silica from the wear of the pebbles. Of the concentrates, the entire product will pass a 200-mesh screen.

The overflow of the Dorrs passes into two Callow de-watering-cones, the spigot-product of which is distributed over 10 amalgamating-plates similar to the coarse amalgamating-plates previously described. From the plates the pulp flows into launders, thence into a 6-in. pipe, 37 ft. long, having a fall of 0.75 in. per foot, which conveys the pulp directly to the lower or cyanide building.

In the lower building the pulp is received into a wooden distributing-box, from which it flows through two Pierce amalgamators into four 8-ft. Callow cones. The spigot-product from these cones discharges into four similar ones placed lower than the first set.

The spigot-product from the lower cones enters one of four Pachuca tanks, where it receives a preliminary treatment of 3 hr. agitation in a solution containing 2 lb. of lime per ton (0.1 per cent.), after which it is allowed to settle and the clear solution is decanted. The filling, agitating, settling, decanting, and discharging of a 25-ton charge of concentrates, which includes 46 tons of lime solution, requires somewhat less than 24 hr. This preliminary treatment saves in the subsequent treatment at least 1 lb. of cyanide per ton of concentrates.

The overflow lime-water from the Callow cones enters the

same sump with the decanted lime-water from the preliminary treatment, and is pumped by an Aldrich triplex 7- by 9-in. electric pump into a reservoir of 75 tons capacity situated in the upper building. The thickened pulp, ranging from 1.8 to 2.2 specific gravity, is drawn into one of eight Pachuca tanks, where it is given the cyanide treatment.

All Pachuca tanks in the mill are 10 ft. in diameter and 30 ft. high, with 60° conical bottoms, Fig. 6. When filled to the level found best for agitating (which is 6 in. below the top of the central column), each tank holds a volume equivalent to 51 tons of water. This is equal to the regular charge of 30 tons of concentrates with 40 tons of solution, although as high as 40 tons of concentrates have been treated as one charge without any difference in extraction-results.

The floors under the Pachucas, as well as all other floors in the building, are of smooth concrete sloping to a central sump, supplied with small pumps to return any escaped solution or pump to the proper tanks.

The first cyanide treatment consists of 8 hr. agitation in a 2-lb. (0.1 per cent.) cyanide solution; either potassium or the mixed cyanides being successfully used. Alkali is kept at 1.25 lb. (0.063 per cent.) of lime (CaO) per ton of solution. Lime is added during the treatment if the titrations show below that figure; 18 hr. is allowed for settlement and decantation of this solution.

Decantation takes place through a flexible hose, which is made as follows: Canvas coated with tar is wrapped around pieces of old boiler-tubing 3 in. in diameter and 4 in. long, spaced 0.75 in. apart. The canvas between the short lengths of tubing is wrapped with wire, making the diameter of these spaces slightly smaller than that of the tubing, thus insuring flexibility as well as avoiding the shifting of the tubing. Attached horizontally to the top of the flexible hose is a 3-in. slotted pipe. In operation this slotted intake floats by the aid of two adjustable air-cylinders. The arrangement of these cylinders is such as to allow of the vertical adjustment of the intake-pipe to any depth of submergence desired.

The long settlement allowed, with the excessively fine condition of the concentrates, their high specific gravity, from 4.6 to 5.0, and the high alkalinity of the solution, leaves a 30-ton

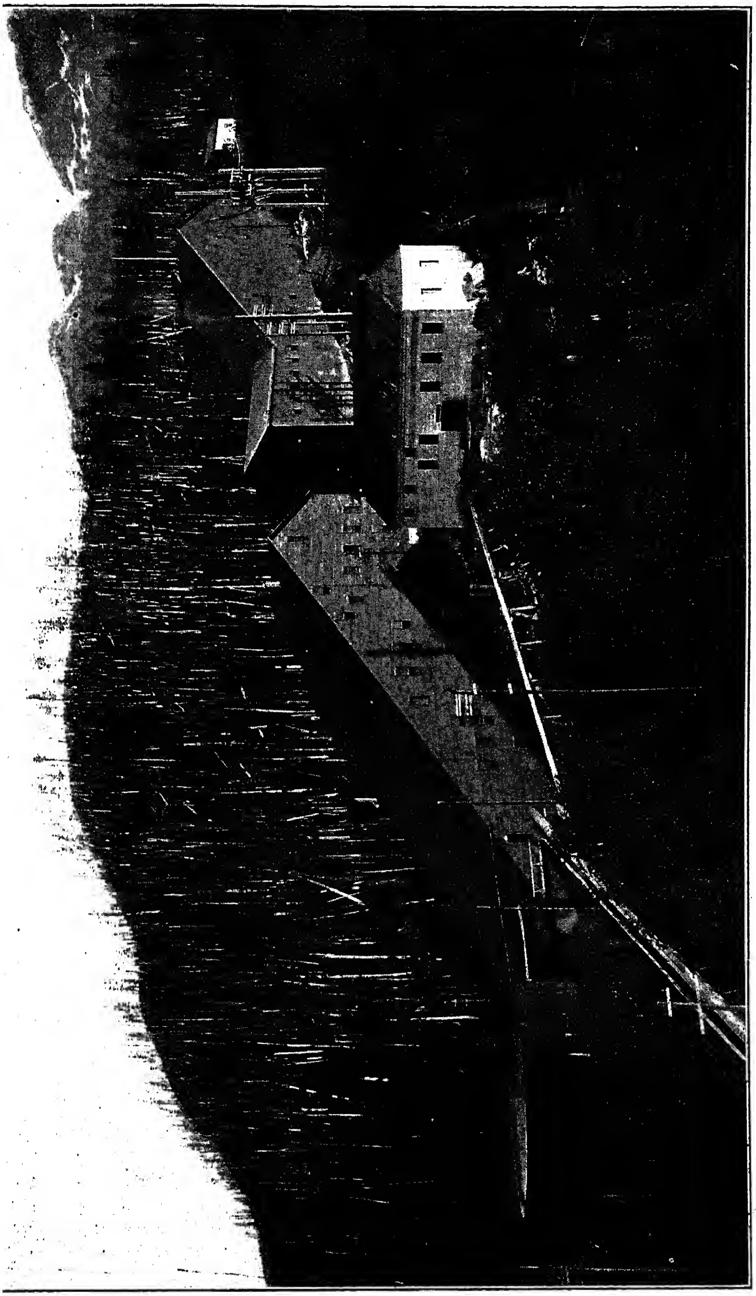


FIG. 3.—CYANIDE-PLANT OF ALASKA-TREADWELL GOLD MINING CO.

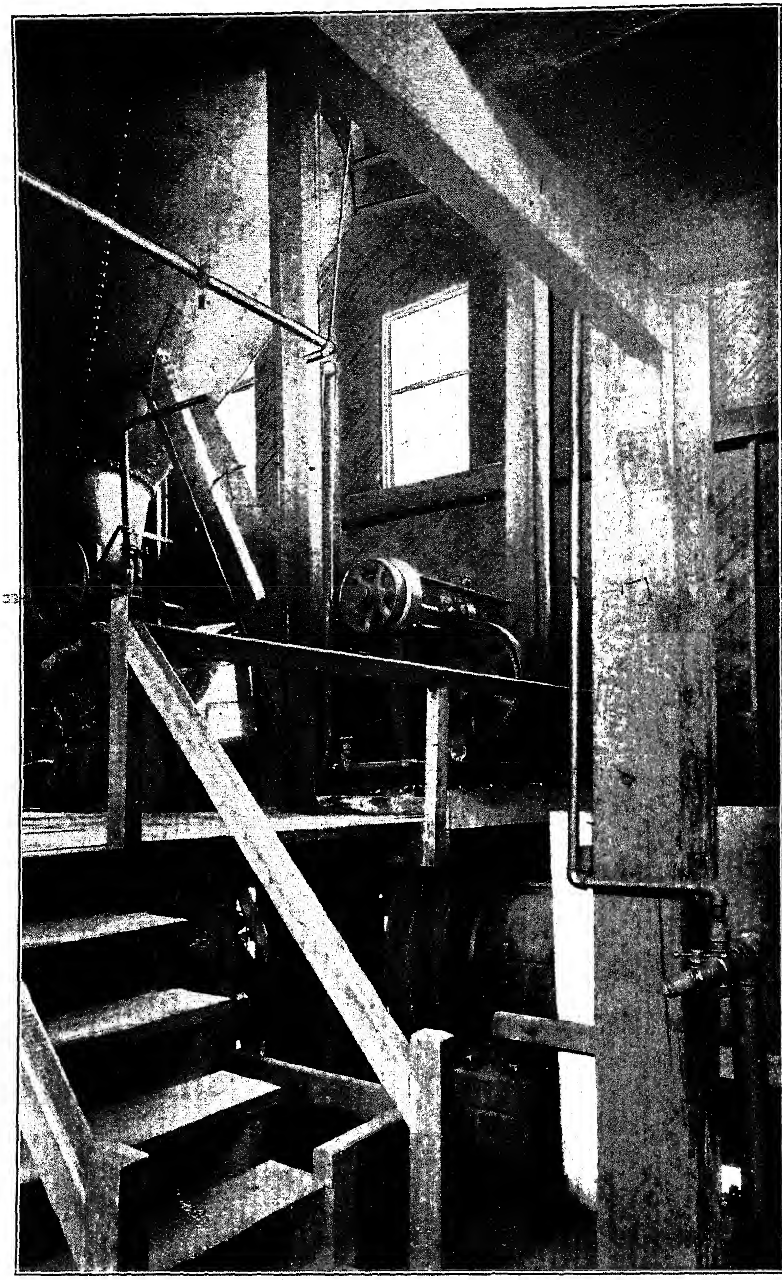


FIG. 5.—CONICAL CONCENTRATES-BIN EMPTYING INTO DORR CLASSIFIER, WHICH IN TURN FEEDS INTO SPIRAL OF TUBE-MILL.

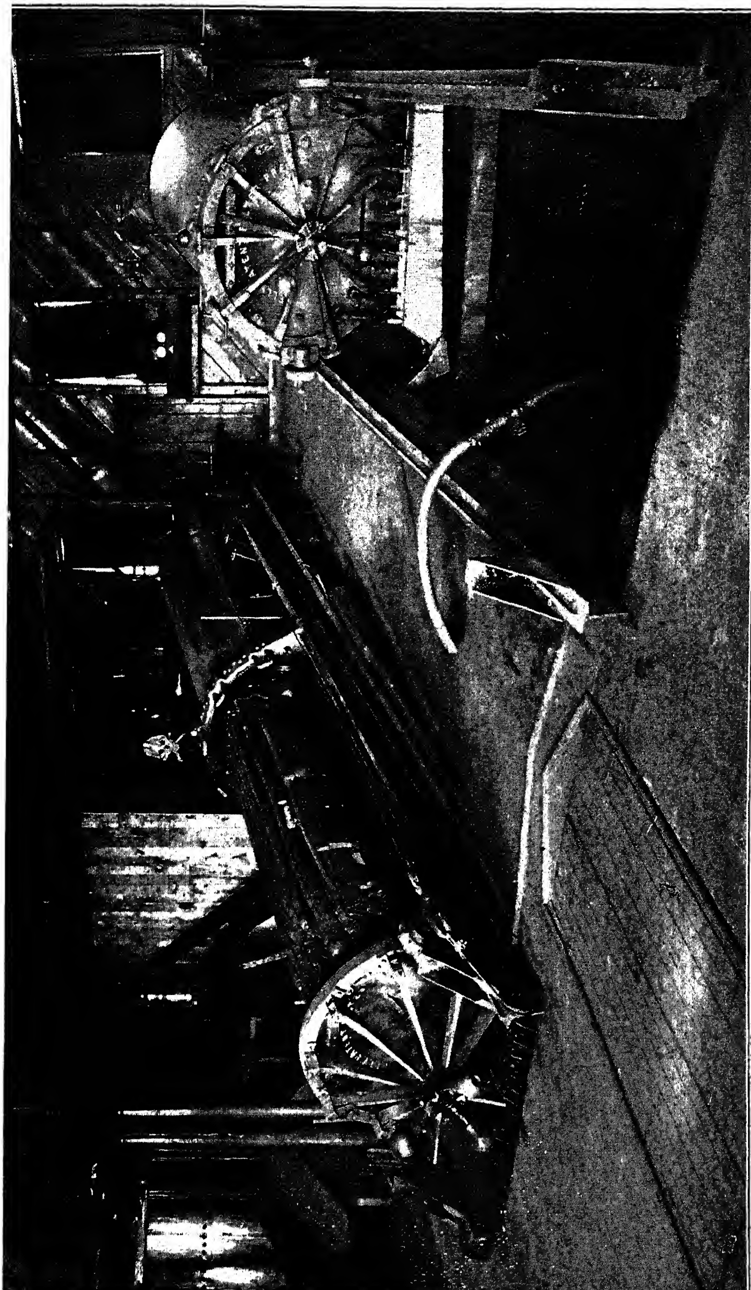


FIG. 7.—KELLY FILTER-PRESSES.

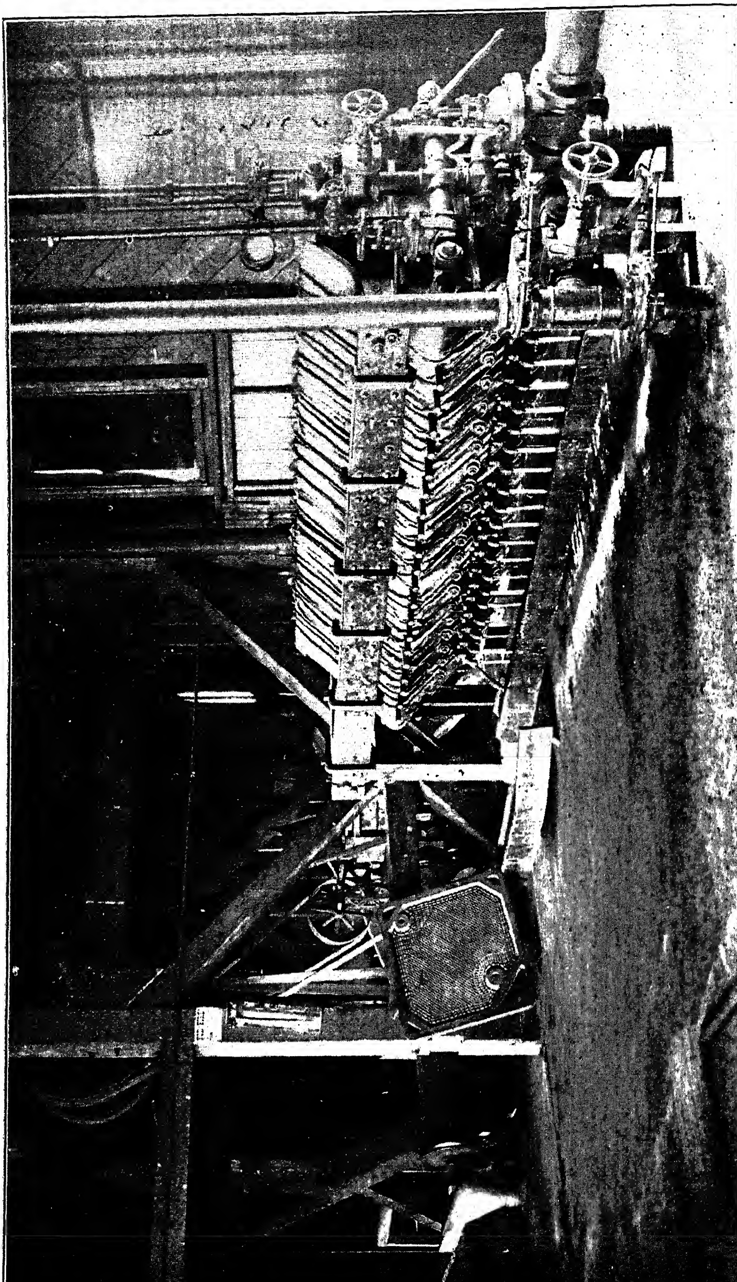


Fig. 8.—Treadwell Sluicing Clarifying-Filter in Operation.

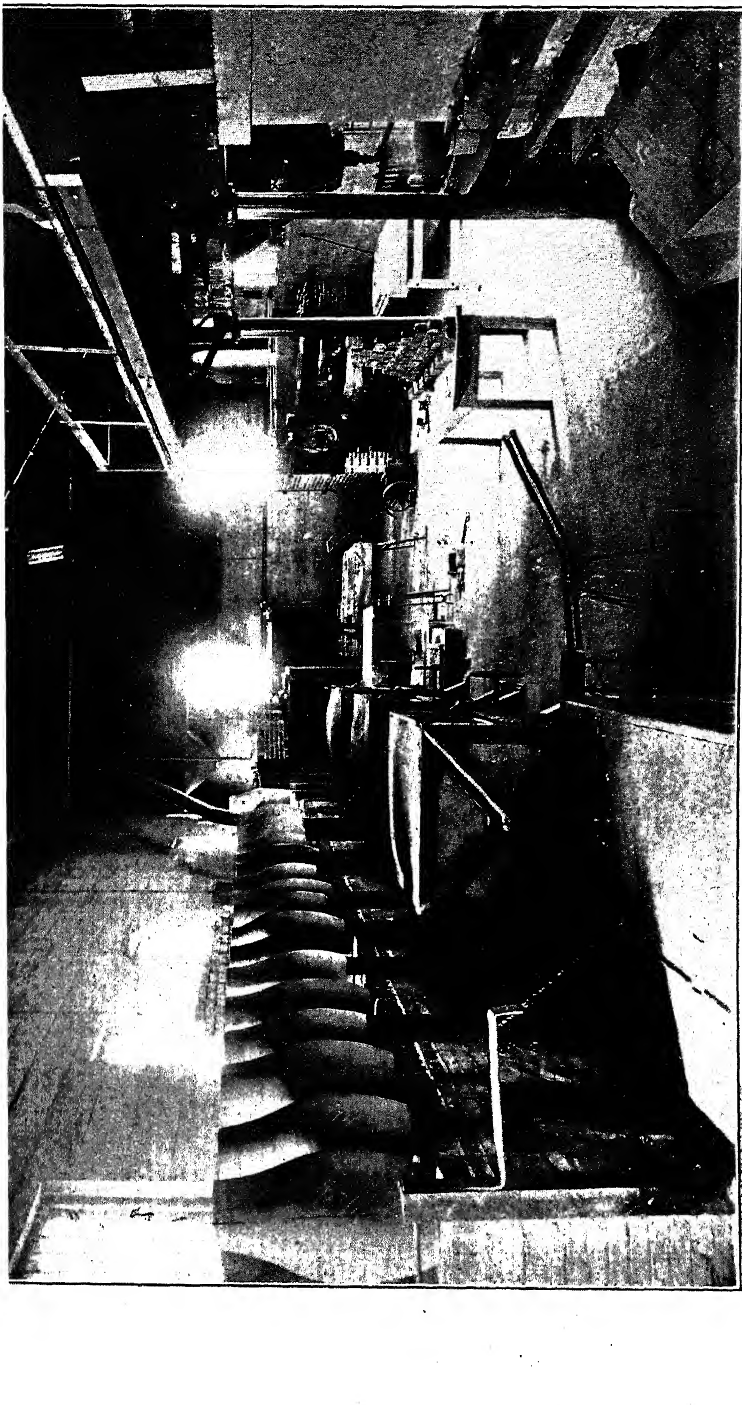


FIG. 9.—REFINERY-ROOM OF ALASKA-TREADWELL GOLD MINING CO.

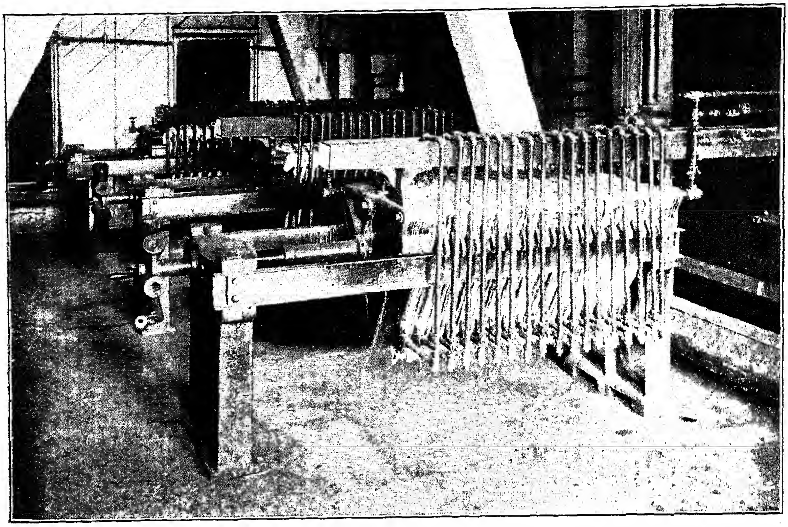


FIG. 10.—INSTALLATION OF MERRILL GOLD-PRESSES, SHOWING METHOD ADOPTED TO PREVENT THE DRAINING OF PRESSES.

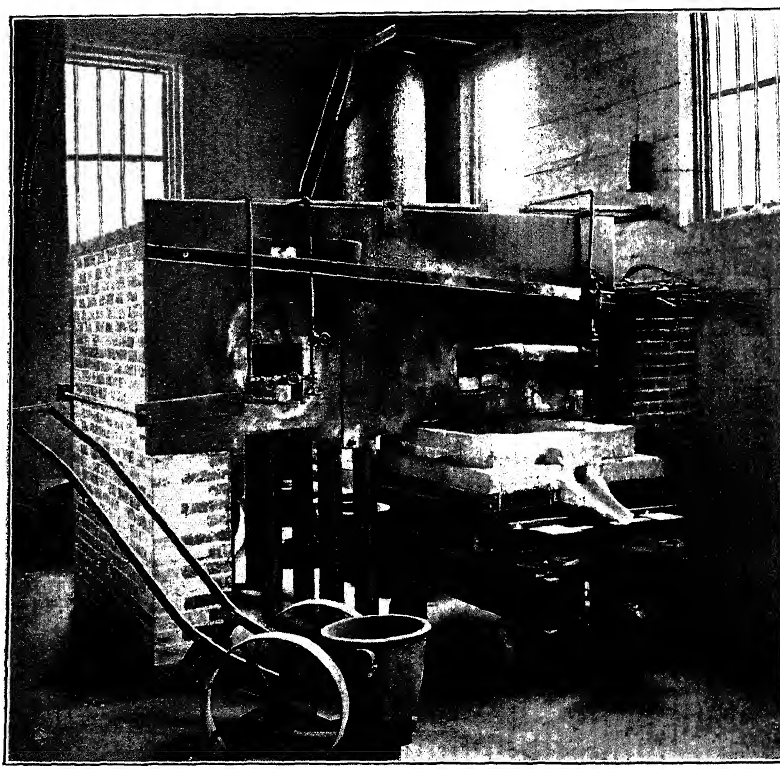


FIG. 11.—CUPELLATION-FURNACE IN USE AT ALASKA-TREADWELL, SHOW-

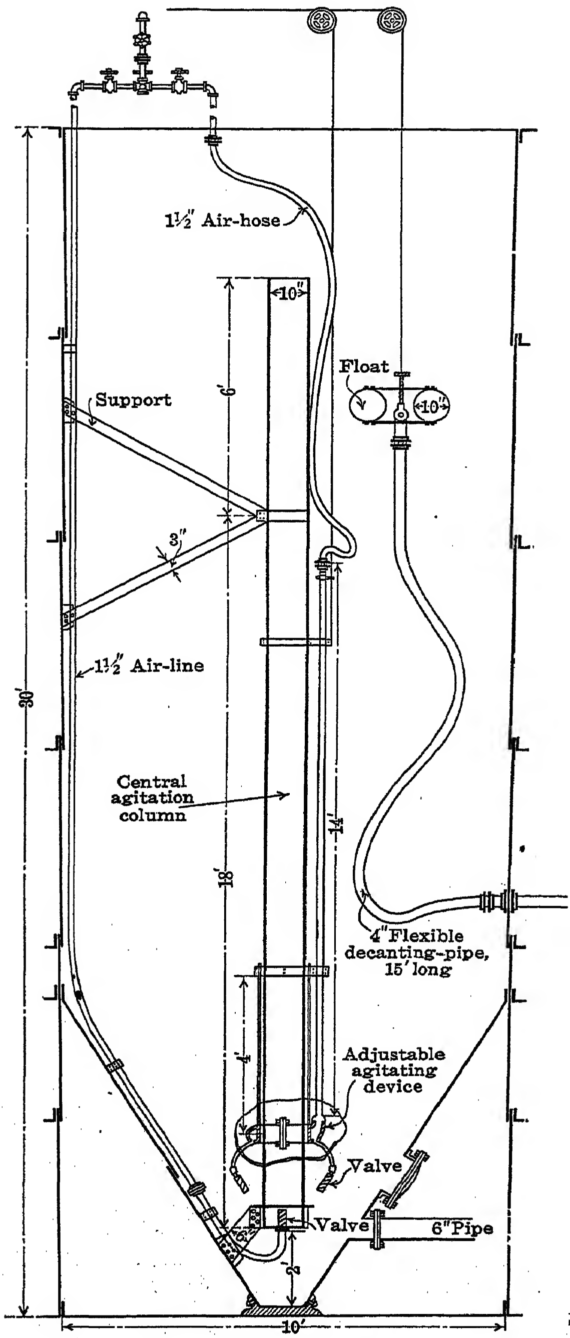


FIG. 6.—PACHUCA TANK.

packed mass in the bottom of the Pachuca. This is brought into thorough agitation within 15 min. by a device designated as the "spider," which is an adjustable hollow annular casting with radiating fingers, the whole encircling the central agitation-column.

When the charge is to be put into agitation the spider is lowered by a small hand-windlass until it rests on top of the settled charge. Air is then turned through the fingers, and at the same time the solution for the next treatment is run into the tank. The device rapidly bores its way to the bottom of the Pachuca, leaving a boiling, churning pulp above, and clearing the way to the bottom opening of the central 10-in. agitating-column. As soon as this is opened and air has been admitted to the inner-pipe the spider is raised from the tank and full agitation of the charge proceeds.

The second cyanide treatment of the charge is with solution drawn from the barren-solution storage-tanks or the wash-solution storage, the cyanide strength being 1.5 lb. (0.075 per cent.) per ton of solution. After 2 hr. of agitation the air is shut off and almost immediately decantation is started. This decanted solution is pumped directly on to an incoming fresh charge, being strengthened in cyanide as it enters the tank, and becoming the first cyanide solution for the new charge.

This cycle in handling solution—barren to wash-solution, then to second cyanide treatment at 0.075 per cent. of cyanide, then to first treatment at 0.1 per cent. of cyanide, thence to precipitation and back to barren—gives at each step just the conditions best suited for that step, and is very satisfactory in practical operation.

The settled pulp after the second decantation has a specific gravity of 1.8, and is readily agitated by means of the spider, and then discharged into the pulp-storage tank by a Byron Jackson 4-in. centrifugal pump, from which it is drawn to the Kelly filter-press. This thick pulp holds in suspension the sands which would settle through a lighter medium.

The storage-tank is conical bottomed, 15 ft. in diameter, and situated at such an elevation that a static pressure of 30 lb. per sq. in. is exerted at the filter-presses. The pulp in the tank is kept in constant circulation by an air-lift, drawing from the conical bottom and carrying the pulp down close under the

filter-presses and back up again over the top of the tank. The pulp as pumped from the Pachuca tanks enters the bottom of this same line, and the whole is thus kept in suspension and circulation past the presses, into which it is intermittently drawn for filter-treatment. Fig. 7 shows the Kelly filter-presses installed to work under a gravity head as described.

Above the pulp-storage tank is placed a similar tank for the storage of wash-water, from which a hydrostatic pressure of 25 lb. per sq. in. is obtained at the presses. This solution is kept in circulation, using the same method as applied to the pulp. The higher gravity of the pulp in the lower tank results in a greater pressure at the presses than that obtained from the wash solution, although the latter carries a higher head.

Filtering is done in two type 1 B Kelly presses. By opening valves in the circulation-lines directly under each press it is filled with either pulp or wash-solution as desired. The excess pulp or wash-solution from the press-cylinder is returned into its proper line by displacing with compressed air admitted into the cylinder. The amount of wash given depends upon the comminution of the concentrates, the usual pulp being washed with 0.5 ton of solution per ton of concentrates. The cake formed during decantation of the first-treatment solution, being very fine slime and more impervious to wash-solution than the regular pulp, is given 1 ton of wash per ton of concentrates.

When filling the press, the contained air is allowed to escape through an overhead pipe attached to the highest point of the press-cylinder. The change in sound of the exhaust indicates to the pressman when the press is full. After drying the cake with compressed air until it contains not more than 10 per cent. of moisture, the press is opened and the cakes shaken off with wooden paddles, and then sluiced with water to the tailings-dam.

A distributor below the press-launder sends the gold-solution to two gold-sumps and the wash-solution to the two wash-solution storage-tanks. These four tanks, as well as a clarifying-tank which is in the same group, are built of 3-in. redwood, 15 ft. in diameter by 16 ft. deep, and each holds 75 tons of solution.

The wash-solution is pumped to a Pachuca tank as needed, becoming a second-treatment solution. From the gold-tank

the solution is drawn into the clarifying-tank, in which are suspended vertically six canvas filter-leaves, all connected to the suction of a triplex 7- by 9-in. Aldrich electric pump, used exclusively for pumping gold-solution through the precipitation-presses. A traveling-belt, driven by ratchet-gears and a pair of eccentrics connected to the pump-drive, feeds zinc-dust into a cone. Here the dust is emulsified with a small stream of gold-solution tapped from the discharge-column of the same pump, and is then drawn into the suction-line. An automatic float in the cone prevents the introduction of air into the pump-suction.

The pump raises the solution with the zinc-dust to the upper part of the building and forces it through two 36-in. triangular, 16-frame Merrill presses, Fig. 8. An average of 145 tons of solution is precipitated daily, with a consumption of $\frac{1}{8}$ lb. of zinc-dust per ton of solution, equivalent to 0.86 lb. of zinc-dust per ton of concentrates. The average strength of solution before precipitation is 1.25 lb. (0.0625 per cent.) of cyanide; 1 lb. (0.05 per cent.) of lime, and $9.50 (9.2 dwt.) gold. The barren or precipitated solutions are kept at 10 cents (2.3 grains), or less, gold per ton, and are used for wash-solution or returned to the Pachuca tanks, as desired.

The Merrill presses are opened when filled or when the pressure exceeds 25 lb. per sq. in. Forcing the solution through at higher pressures caused a mechanical loss of precipitate through the canvas. The precipitate is dropped from the press-frames into steel pans and lowered by an electric elevator to the floor below, and thence conveyed by trucks through a concrete passage into the refining-room.

IV. Cyaniding without Amalgamation.

On account of the work required to look after and collect the amalgam, as well as the greater danger of amalgam-loss from the pipe-lines, launders, etc., the plates were removed after the first three months' run, and the whole product is now being cyanided directly without amalgamation.

In order to handle the larger amount of solution made necessary when grinding in cyanide solution, two 1,800-ton steel tanks have been erected, one above and one below the plant. All the precipitated or barren solution flows by gravity from

the precipitation-presses to the lower tank. This solution, having an average value of $0.08 in gold, 1.14 lb. of cyanide, and 1.70 lb. of lime per ton, is pumped to the second of these tanks, which is situated 25 ft. above the mill-bins, and acts as the mill-reservoir.

Thus at no time is there any cyanide solution run to waste, the solution discharged as moisture in the tailings, plus that absorbed or evaporated in the mill, compensating for that received as moisture in the concentrates delivered to the bins.

All the solution used in grinding and classifying is drawn directly from the mill-reservoir. The overflow of fine pulp from the back of the Dorr classifier flows at once to the Callow tanks in the lower building, the spigot-product of which empties into one of the 12 Pachuca tanks for treatment.

The specific gravity of the pulp as it enters the Pachucas is 1.5, or a ratio of 1 of concentrates to 1.18 of solution. The charge is agitated for 8 hr., the necessary cyanide and lime being added to bring the cyanide-content of the solution to 1.5 lb. (0.075 per cent.) and the lime-content to 2 lb. (0.1 per cent.) per ton. After agitation and settlement, the clear solution is decanted to the gold-tank through the clarifying-press described later, and a fresh charge of barren solution, the same as that used in the grinding, is drawn from the mill-reservoir, brought to the same strength as the previous treatment, and the charge agitated for 4 hr. This is then settled and the solution is decanted. Both solutions decanted from the agitators, together with the overflow from the Callow settling-tanks previously mentioned, are drawn by gravity through the clarifying-press before emptying into the gold-tanks.

The settled pulp in the bottom of the Pachuca tanks, having a specific gravity of 2, is then agitated by means of the spider and pumped to the pulp-storage tank, from which it is drawn to the Kelly presses for filter-treatment.

This method of operation, depending upon the one barren solution for all purposes, keeps the gold-content of the solution to the lowest possible value, which, although contrary to the usual practice, is the object sought in this mill.

The solution overflowing from the Callow settling-tanks (containing gold, $10; cyanide, 1 lb.; and lime, 2 lb. per ton) flows by gravity through a special clarifying-press built in the

Treadwell shops, the same as receives the decanted solution. This press is of the ordinary plate-and-frame type, yet with a series of ports or channels so arranged as to allow of discharging or sluicing-out a cake without the necessity of opening the press. This sluicing-out press consists of 20 square frames, each 3 in. thick, with the corresponding plates 1 in. thick. The upper and two side-channels extending through the press have small holes opening into the frame side of the leaf. The upper small channel allows the introduction of compressed air behind the leaves. The lower triangular channel connects with a 6-in. sluicing-out pipe. The press, with connections, is shown in Fig. 9, with one of the plates standing to the left.

To discharge a cake, water is introduced at the back end of the press through the large triangular opening on the bottom, and flows through the underside to the discharge end, where it empties into the launder leading to the tailings-dam. With this passage-way clear, compressed air is introduced through the port-holes on the plate side of the leaf. The plate corrugations being depressed 0.5 in. leaves a concave surface, in which the cake forms. The air now being introduced behind the leaves, by a series of separate knocks or bumps causes the cakes to drop off into the sluicing-out channel, where they are carried away by the stream of water.

For the final washing of the leaves, water is introduced through the three upper channels, and, passing through the tapered holes, is sprayed on the two filter-cloths, which bag together by reason of the compressed air introduced from the plate side.

The method of feeding the zinc-dust has been changed somewhat from that originally installed. The reasons for these changes were to create a more even feed of zinc, to do away with the air previously used in the emulsion-cone, and not only to break up any lumps, but to brighten the zinc and grind it even finer. To do this, the drive from the zinc-belt was taken from the Aldrich pump to a small counter-shaft, which was, in turn, belted to a worm-gear for the drive of the zinc-belt, the belt discharging its zinc directly into a small tube-mill 6 ft. long, made from 10-in. pipe, the cast-iron caps of which were turned to run in rollers. This tube is filled with rods of cast zinc 2 in. in diameter. These rods not only grind the zinc to

a more uniform product, but may themselves aid precipitation to a slight extent.

Considerable annoyance is occasioned by the clogging of the cloths in the Merrill gold-presses and by the accumulation of precipitate in the entire line from the zinc-feeder and pump to the presses. Filter-cloths of several kinds—heavy duck at 31 cents per yard, various grades of drilling at from 9 to 15 cents, and muslin sheeting at 7 cents—have been tried. The lightest and cheapest muslin is now in use, with results no worse than obtained with the more expensive grades.

From the moment of contact of the zinc-dust with the gold-solution trouble is caused by the slimes or precipitate incrusting everything touched. The interior of the pipes gradually becomes smaller in area, even though the solution is driven through at a constantly increasing velocity. After three months' use a 6-in. pipe of 28 sq. in. area was so filled with caked precipitate that only a triangular opening of 4 sq. in. remained. From 80 ft. of this pipe, $25,898 was recovered.

Being desirous of operating the Merrill presses more or less intermittently without the necessity of each time closing the cocks to retain the solution, which if allowed to drain not only oxidizes the zinc, but causes the precipitate when the pressure is removed to settle in a mass at the bottom of the press-frames, consequently not allowing the greatest amount of solution to pass through the unoxidized zinc, the discharge-cocks were removed from the plates, and open pipes discharging into a launder on top of the presses were substituted, as shown in Fig. 8.

The result of the several changes is a more uniformly low tail solution, with the consumption of less zinc, while the gold-value of the precipitate has been raised from $15 to $25 per pound; hence a corresponding lowering of refining-charges.

V. The Refinery.

The refinery adjoining the mill is 30 by 76 ft. in area; constructed of reinforced concrete with steel-truss roof covered with corrugated iron, shown in Fig. 10. The precipitate entering the refinery is crushed through 0.5-in. screen, made up into lots of from 1,000 to 1,200 lb., weighed, sampled, and charged into one of two redwood tanks, 8 ft. in diameter and 9 ft. deep,

with a conical bottom and lined with sheet-lead. The tanks are built on the plan of a Pachuca tank, with a central column of wood fitted with lead pipes carrying steam and compressed air for heating and agitating the solutions.

In these tanks the precipitate is treated with acid to dissolve out the zinc, lime, etc. About 1 lb. of 66° sulphuric acid is required per pound of precipitate, and is added in the following manner: About 2 tons of water is introduced into the tank, steam turned on, and the water brought to the boiling-point. Air is turned on in the central air-lift, and the acid-valve opened. The acid flows in by gravity, while the precipitate is shoveled in at the rate of 2 lb. of precipitate to each pound of acid. When all the precipitate and from 50 to 60 per cent. of the acid have been added, the acid-valve is closed and the charge agitated until the acid is entirely neutralized, which generally occurs within 30 min. The tank is then filled with water, and the charge allowed to settle for about 2 hr., after which the clear solution is siphoned off into a filter-tank. The latter is 8 ft. in diameter and 4 ft. deep, having a false bottom of 1-in. strips, placed 12 in. from the bottom of the tank and 1.5 in. apart. The strips are covered with heavy iron screen, 1-in. mesh, on which is a bed of burlap 1 in. thick, one thickness of mill blanket, one thickness of light canvas, and a bed 1 in. thick of quartz sand screened between 20- and 80-mesh. The sand is divided into sections of 8 by 10 in. by a light wooden frame, covered by a single thickness of drilling, the latter forming the working-surface of the filter. The solutions filter freely through this medium, the clear filtrate being run into one of three storage-tanks, where it is held until a sample has been assayed, and then run to waste through a series of zinc-boxes. All solutions and wash-waters from the refinery are disposed of in this way.

After decanting the first acid, the precipitate in the tank is given two washes of boiling water. Just enough water to enable the charge to be agitated is then added, and the remainder of the acid run in rapidly. This gives a solution containing from 15 to 18 per cent. of acid, agitation being continued until the acidity ceases to decrease, which usually leaves about 1 per cent. of free acid. The tank is then filled with water, settled

and decanted as before. This solution, containing from 50 to 75 lb. of free acid, is at present run to waste.

The charge now receives three or four washes of boiling water followed by washes at about 30° C. temperature, until the wash-water gives no reaction for sulphates with barium chloride, which is generally after 15 washes. After decanting the last wash, the charge is sluiced through a valve in the bottom of the tank on to the filter, which has been thinly covered with silica sand to aid filtration, where the excess water is removed by means of a vacuum-pump. The slimes are removed to a large wrought-iron pan, placed upon a 4- by 8-ft. steam-table, inclosed by a sheet-iron hood. When nearly dry but still damp enough to prevent dusting, the slimes are rubbed through a 0.5-in. screen, weighed and sampled, the weight of the acid-treated product being from 25 to 33 per cent. of that of the original precipitate. Each lot of precipitate is analyzed before and after the acid treatment, which enables a close calculation to be made of the amounts of fluxes required for the monthly melting.

The percentages of the principal substances contained in an average analysis of the precipitate before and after acid treatment are:

	Before. Per Cent.	After. Per Cent.
Au	5.08	17.34
Zn	42.93	5.15
Pb	8.08	20.09
Cu	6.19	14.28
Cao	10.51	1.89
Fe	1.10	0.52
S	1.41	7.62
Insoluble,	3 48	22.85

The high percentage of insoluble after treatment is due to the silica added to the lot just before filtering.

At the end of the month the various lots of acid-treated precipitate are united and the various fluxes added. The melting is done in a specially-constructed oil-burning furnace (Fig. 11). For melting purposes the furnace is fired with a reducing flame. The crucible or hearth used for the melting is 4 by 3.5 ft., lined with either magnesite brick or fire-clay, according to the fluxes used. This hearth is placed on a steel car and run

under the furnace. Jack-screws, operated by hand-wheels at the four corners of the car, allow of raising the hearth to form the furnace-bottom.

From the fire-box at one end of the furnace the heat is drawn across the top of the charge, being reflected downward by the arch roof and the down-draft to a dust-condensing chamber. The furnace is charged with precipitate hourly, the slag and lead-bullion being tapped off intermittently from opposite sides of the hearth. The month's clean-up, amounting to 1,450 lb. of acid-treated precipitate, or a total charge, including fluxes, of 2,600 lb., is melted on this hearth in 36 hr., and requires the attention of but one man per shift.

A typical mixture of fluxes is:

	Pounds
Acid-treated precipitate,	100
Borax glass,	22
Sodium carbonate,	25
Old slag,	80
Iron-turnings,	15
Powdered graphite (old retorts),	3

Such a charge will produce about 35 lb. of metal, from 10 to 15 lb. of matte, and from 160 to 180 lb. of slag.

From 150 to 300 lb. of high-grade copper-matte are produced each month. This matte is roasted and allowed to accumulate until there is sufficient to make up a charge, when it is mixed with litharge, fluxed, and melted to produce lead-bullion, which is the work-lead used for the removal of copper in cupellation.

After melting either precipitate or matte, the slag is tapped into conical pots holding about 200 lb., with a tap 4 in. from the bottom, through which the molten core is drawn off. The shells, containing most of the metallic values, are dumped, crushed, and used in fluxing a later charge. The cores, constituting 75 per cent. of the total slag, are sampled, sacked, and stored for shipment to the smeltery.

The cupellation is done on a limestone test the same size as the melting-hearth, it being run under the furnace on the car previously described. For cupellation the furnace is fired with an oxidizing flame, while free air is introduced over the test by means of a connection from a compressed-air main through a needle-valve discharging into the open end of a

4-in. pipe. This produces low-pressure air, which is introduced into the furnace on the opposite side from which the molten litharge is tapped off.

The fine bullion resulting from this cupellation is drawn off and remelted in Faber du Faur tilting-furnaces into bars of 1,000 oz. each. The average fineness of the cupelled gold is 880.

The retorts of the Faber du Faur furnaces are supported on two 1.5-in. iron pipes built into the furnace, through which cold water is kept circulating. These pipes have proved very satisfactory.

VI. Costs.

In conclusion, the cyanide-plant has now been in operation one year, using the machines and equipment originally installed, with the exception of the abandoned amalgamation-plates, the substitution of larger tube-mill motors, and the addition of the "Treadwell" clarifying-press, with results summarized in Table VI.

For the last month, ending Aug. 15, and not included in cost-sheet, 2,010 tons were treated, at a cost of $2.8764 per ton, and an estimated extraction of 97.025 per cent., as compared with the experimental estimates of $3.25 per ton and 96 per cent. extraction.

TABLE VI.—*Costs of Cyanide-Treatment, Alaska-Treadwell Gold Mining Co.*

Treatment	Regrinding and amalgamation, followed by agitation in cyanide solution.						Regrinding in cyanide solution, followed by agitation in cyanide solution. (No amalgamation.)						Regrinding in cyanide solution, followed by agitation in cyanide solution. (No amalgamation.)					
Period covered	Sept. 16 to Dec. 20, 1910, trial run and first adjustments of the plant. Average head value: $54.6174. Average tailing value: $1.7589. Estimated extraction: 96.80 per cent. Tons treated: 4906.1						Feb. 16 to May 15, 1911. (Plant idle Dec. 21, 1910, to Feb. 15, 1911.) Average head value: $46.9360. Average tailing value: $1.7990. Estimated extraction: 96.17 per cent. Tons treated: 4620.6						May 16 to July 15, 1911. Average head value: $55.4899. Average tailing value: $1.8892. Estimated extraction: 96.60 per cent. Tons treated: 4080					
	Costs Per Ton Treated.						Costs Per Ton Treated.						Costs Per Ton Treated.					
			Supplies.						Supplies.						Supplies.			
Operation.	Labor (d)	Electric Power, Compressed Air, Steam Heat. (c)	Item.	Cost.	Sundry Supplies	Totals	Labor (e).	Electric Power, Compressed Air, Steam Heat. (c)	Item.	Cost	Sundry Supplies	Totals.	Labor (e)	Electric Power, Compressed Air, Steam Heat. (c)	Item.	Cost	Sundry Supplies	Totals.
Grinding	$0.2033	$0.6179	Pebbles 8.05 lb.	$0.1182	$0.0419	$0.9843	$0.1512	$0.4679	Pebbles 22.19 lb.	$0.2686	$0.0061	$0.8939	$0.1196	$0.1279	Pebbles 13.55 lb.	$0.1776	$0.0059	$0.4810
Amalgamation	0.2324	0.0009	Mercury 3.3 oz.	0.1516	0.0066	0.3915												
Cyaniding	0.3291	0.1378	Lime 19.78 lb.	0.1978	......	....	0.2646	0.1007	Lime 12.77 lb.	0.1226	...	....	0.2093	0.0396	Lime 7.06 lb.	0.0649		
			Cyanide 2.46 lb.	0.5163	0.0201	1.2011			Cyanide 2.86 lb.	0.5665	0.0251	1.0795			Cyanide 2.49 lb.	0.5330	0.0018	0.8486
Filtering	0.3389	0.0791	Filter cloth	0.0162	0.0233	0.4416	0.2646	0.0729	Press-cloth.	0.0109	0.0053	0.3537	0.2093	0.0346	Filter-cloth	0.0038	0.0059	0.2536
Precipitation	0.1549	0.0645	Zinc-dust 0.79 lb.	0.0592	0.0053	0.3001	0.1843	0.0567	Zinc-dust 2.16 lb.	0.1685			0.1458	0.0234	Filter-cloth	0.0103		
									Filter-cloth	0.0204					Zinc-dust 2.09 lb.	0.1700		
									Lead acetate 0.54 lb.	0.0511	0.0131	0.4894			Lead acetate 0.15 lb.	0.0142	0.0209	0.3846
Refining (a)	......	....	....	..	..	...	0.2268	0.0436	Retorts.	0.0619			0.2243	0.0243	Fluxes.	0.0193		
									Sulphuric acid	0.0617	0.0252	0.4192			Sulphuric acid	0.0656	0.0501	0.3836
Analyses, assays, accounting (b)	.....	.. ..	.	.	0.1780	0.1780	0.1269		. .		0.1687	0.2956	0.0718	...	Chemicals	0.0142	0.0176	0.1366
Superintendence	0.1936	....	.. ...	.	. .	0.1936	0.1890		. .	.		0.1890	0.1495		. .			0.1495
Repairs	......	0.0811	.. ..	..	0.0179	0.0990	0.1118	. .	. ..		0.0069	0.1187	0.1121	.	. .		0.0274	0.1395
Totals	$1.4522	$0.9816		$1.0593	$0.2961	$3.7892	$1.5492	$0.7418		$1.3272	$0.2507	$3.8689	$1.2447	$0.2498		$1.0729	$0.1596	$2.7270

(a) During the first period the refinery had not been completed and the precipitate was shipped to a smelter, cost of which is not given.
(b) Assays are made at the central assay office of the combined companies at Treadwell, and charged against the cyanide plant at a proportionate rate per assay.
(c) Electricity, compressed air, and steam are purchased from central power-plants, at cost of generation.
(d) Labor, first period, 20 men at $3.75 average wage.
(e) Labor, second and third periods, 22 men at $3.784 average wage. (18 men in mill, 4 men in refinery.)

The Parral-Tank System of Slime-Agitation.

BY BERNARD MACDONALD, GUANAJUATO, MEXICO.

(San Francisco Meeting, October, 1911)

Introduction.

Of the treatment of the slime-pulp of gold- and silver-ores by cyanidation, agitation is an essential part. When prepared for treatment, this pulp, consisting of ore reduced to such fineness that approximately 80 per cent. of it will pass through a 200-mesh screen, is mixed with a certain proportion of water, carrying in solution the quantity of cyanide (KCN) and other chemicals required.

The water-constituent of the pulp thus prepared usually ranges from 1 to 2 parts by weight to 1 of the dry ore. Thus constituted, the pulp is charged into treatment-tanks, the shape and capacity of which vary, according to the quantity of pulp to be treated daily and the method of agitation to be employed. Tanks have no other function in a cyanide-plant than that of being economical and convenient containers or receptacles for holding the pulp, solution, or water used in the operations.

It is by agitation that the solids in the pulp charged into the tanks are kept in suspension in, and mixed with, the solution in the proper proportions required for the treatment. If the proper mixture of solution to solids be determined to be 2 to 1 by weight (which is, approximately, 5 to 1 by volume), this proportion should be maintained in every part of the charge; that is, each solid particle of the pulp, whether it be of 180- or 400-mesh size, should be surrounded by five times its own volume of solution throughout the whole period of treatment. The reason is, that the amount of chemicals ascertained to be necessary for dissolving the gold and silver contained in the solids, is held in uniform solution in the water-constituent of the pulp, and, therefore, the determined proportions of solution and solids must be maintained at all times during treat-

ment. If the pulp should be allowed to thicken at the bottom of the tank so that it would contain, by volume, say, only 4 parts of the solution to 1 part of solids, it is plain that there would be present in this part of the tank-charge only four-fifths of the chemicals necessary for the treatment of the solids, while the one-fifth lacking would be present in another part of the tank-charge where it was not required. This principle, the importance of which is not always appreciated in the operation of a cyanide-plant, is the main ground for the necessity of agitation. But, besides maintaining the proper proportional mixture of solution and solids in the tank-charge, agitation is designed to give the required "aeration" to the pulp during treatment.

Means of Effecting Agitation.

In the cyanide-plants built before 1907, agitation was effected in tanks 10 to 12 ft. deep, and ranging in diameter up to 30 ft., by mechanically revolving stirring-arms, assisted by centrifugal pumps drawing the settled pulp from the bottom of the tank and throwing it back on the top of the charge in the same tank. This method was fairly efficient, but expensive in both the construction and the operation of the plant; and it was superseded by pneumatic or air-lift agitation, which proved to be at least equally efficient, and much more economical.

The method of air-lift agitation which came into general use in cyanide-plants is known as the Pachuca-tank system. The superior economy of air-lift agitation and the energy of the patentees of this system soon brought this method into popularity, and most of the recently constructed cyanide-plants have adopted it.

Analysis of the Pachuca Tank and Its Operations.

Fig. 1 is a sketch of the Pachuca tank and its pipe-equipment. Beside it is shown a Parral tank of equal holding-capacity, Fig. 2. The Pachuca tank is a tall cylinder with a conical bottom. In the center of the tank is fixed the air-lift tube, which, commencing about 18 in. from the apex of the bottom, extends to within a few inches of the top of the tank. The diameter of this tube is proportioned to the diameter of the tank as 1 to 12 approximately.

In Fig. 1, *AA* are the sides of the tank; *BB* is the air-lift tube; *CC*, the pipe which delivers the compressed air into the bottom of the air-lift tube; *D*, the foot-rest which holds the compressed-air pipe in the center of the air-lift tube; *EE*, an auxiliary compressed-air pipe used for delivering compressed air at the bottom of the tank, to keep the pulp in agitation while the charge is being received; *FF*, a system of pipes extending radially from a hollow "bustle" or distributor attached

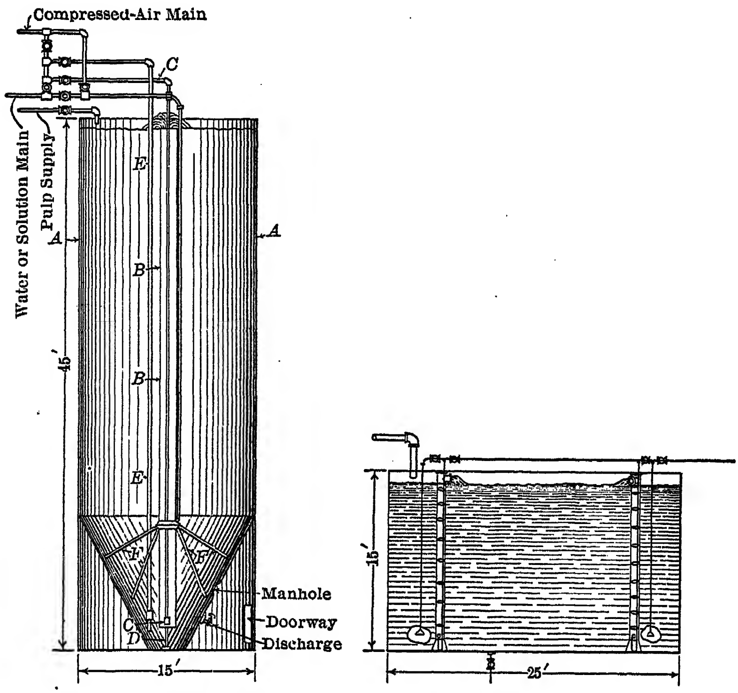


Fig. 1.—Pachuca Tank. Fig. 2.—Parral Tank.

FIGS. 1 AND 2.—PACHUCA AND PARRAL TANKS OF APPROXIMATELY EQUAL HOLDING-CAPACITY.

to the air-lift tube, to which is connected a feed-pipe leading from the air-main at the top of the tank, through which feed-pipe compressed air or solution under pressure may be turned into the bottom of the tank, to assist in agitating the pulp while the tank is being charged, or, in case of packing, to restore the pulp to a fluid consistency so it can be moved through the air-lift tube.

The compressed-air, high-pressure solution, and pulp-charging mains for the pipe-connections are shown at the top of the tank. It should be noted that the end of the compressed-air pipe, *CC*, is capped, and, for a length of about 7 in. next to the cap, is perforated by a number of small holes through which the compressed air escapes into the air-lift tube. To prevent the pulp from entering these holes and choking the pipe, when the compressed air is shut off, a tight-fitting rubber stocking or tube is drawn over the holes and clamped to the pipe above them. When the air is on, this stocking expands and the air flows underneath it and escapes at its lower end, which is left open. When the air is shut off, the stocking closes over the perforations and prevents the pulp from entering them.

In operation, when the tank is receiving its charge from the pulp-charging main, compressed air is turned on through pipe *EE* to keep the pulp in agitation and prevent it from settling in and around the bottom of the air-lift tube.

In case the compressed air fails during the charging of the tank, and the pulp packs so hard around the bottom of the air-lift tube and the rubber stocking as to prevent the operation of the air-lift when the compressed air comes on, air, or solution, or both, may be turned into the auxiliary pipes *EE* and *FF*, to bring back the packed pulp to fluid consistency; and, in case this fails, the tank is provided with a man-hole, shown in the figure, which may be opened, and the packed pulp excavated.

When the tank has received its full charge of pulp, compressed air is turned on in pipe *CC*, which starts the operation of the air-lift tube, and the auxiliary air-agitation pipes are then closed off. By the operation of the air-lift tube, the thick pulp at the bottom of the tank is drawn into and carried up through it, and discharged at the top, where it falls back on the tank-charge and mingles with the thin pulp there.

The transfer of the pulp from the bottom to the top of the tank continues throughout the treatment-period, and preserves the proper proportional mixture of solution and solids. By these means and in this manner, the agitation of slime-pulp is effected by the Pachuca-tank system.

Defects of the System.

That this system was a great improvement over any other previously employed, there is no question; but that it has a number of commercial defects, is also true.

All these defects result from the design of the tank, and the apparatus with which it is equipped, the tank-dimensions being at variance with all the principles governing the object (other than as stand-pipes) for which tanks are employed. The great height and small diameter make the holding-capacity comparatively small, and consequently its cost of construction per unit of holding-capacity, high. The height of the tank and the large diameter of the air-lift tube necessitate a correspondingly high pressure and a large volume of compressed air to effect the transfer of the pulp; and this adds to the cost of agitation.

The pulp transferred through the air-lift tube overflows on the top of the charge, close around the tube, in which relative position the solid particles settle vertically to the bottom, where the steeply-sloping sides of the cone bottom carry them to the intake of the lift-tube, which throws them back again on the top of the charge. Under normal conditions of operation, the air-lift tube turns over the entire charge in a Pachuca tank of standard size in about 15 min. The violence of this operation would not be necessary to keep the pulp in proper mixture; but on account of the tall, narrow tank and the conical bottom, it is necessary, in order to keep the air-lift tube and the air-nozzle from being choked.

The air-nozzle within the lift-tube is a crude mechanical device, expensive to operate and expensive to maintain.

Before the proofs for these assertions are submitted, the principles of air-lift pumping should be reviewed. Those who have never had occasion to investigate the phenomena of air-lift pumping will find the subject fully dealt with in the experiments and conclusions of Dr. Pohle, who obtained a patent from the United States for the use of compressed air in pumping.

From Dr. Pohle's experiments and those made by myself, my understanding is that pumping by compressed air is effected in the manner described below, with due reference to

the conditions of the air-lifting or transfer of pulp in a tank for the purpose of effecting agitation. At the starting of agitation, after the tank has received its charge, the pulp-level is the same within and without the air-lift tube, which extends, say, 3 or 4 in. above the pulp-level. If the pulp has the consistency of 2 to 1 of solution and solids, the pulp-pressure on the bottom of the tank will be 0.54 lb. for each foot in height of tank-charge. The air-pressure for the agitation of such a charge should be 10 per cent. greater, or, say, 0.60 lb. for each foot in height of the charge. When the compressed air at this pressure is turned on in the air-pipe terminating near the bottom of the lift-tube, it flows into the pulp there, which has a pressure of only 0.54 lb. per foot of height. The compressed air, on entering the pulp in the lift-pipe, assumes the form of bubbles; and these, rising through the pulp, immediately unite to make a large flattened bubble which, extending to the sides of the pipe, takes the form of a disk or piston, in which form it rises to the surface, pushing the pulp before it. Rivalry now begins between the pulp and compressed air for the privilege of filling the space vacated by the ascending air-disk. The pulp, endeavoring to restore the hydrostatic equilibrium between the contents of the air-lift tube and those of the tank outside, and aided by its greater volume (due to the disparity of size between the compressed-air and air-lift tubes), rushes past the air-nozzle, holding back for a moment the issue of air. But, immediately, the air, on account of its higher pressure, again succeeds in entering the lift-tube in sufficient quantity to form another air-disk, with the same result as before. Thus by frequent jets of compressed air, alternating with rushes of pulp into the bottom of the air-lift tube, the lifting-operation is effected. The *modus operandi* of the air-lift, as above briefly described, is disputed by some, who hold that the inflow of air is continuous, and that the lifting effect is produced by the formation of a large number of bubbles in the pulp in the lift-tube, which makes it lighter, and, consequently, subject to displacement by the heavier pulp in the tank outside, rushing in at the bottom of the tube, and causing the discharge of the lighter pulp at the top.

A little study will show that this apparently logical reason-

ing cannot account for the operation of the air-lift, for individual bubbles rising through the liquid in the air-lift tube could have no more effect in lessening the hydrostatic pressure at its intake than would so many corks rising through it. On the contrary, it will readily be seen that, were the corks to unite and form disks or pistons filling the pipe, these disks would, on rising through the lift-pipe, carry the intervening pulp upward with them.

It is not improbable, however, that in certain kinds of liquids having great viscosity, the inflow of compressed air would be imprisoned as numerous small individual bubbles, and would in this way form an emulsion of the liquid within the tube, which emulsion, being lighter than the pulp outside, would be lifted or shoved upward by the heavier pulp coming in to displace it. But this condition would not be probable in the case of an ore-slime.

The principal defect of the air-nozzle of the Pachuca tank is the amount of ineffective work that must be done by the compressed air in making its numerous jet-like escapes into the air-lift tube. The superficial area of the exterior of the rubber stocking, that must open and close for each jet of air escaping, is 36 sq. in. at least; and on each inch of this area there is a continuous pressure of 0.54 lb. per foot in height of the tank-charge. As filled in operating, there are 43 ft. of pulp in the tank, making an external pressure of 23.22 lb. per square inch, or a total of 836 lb. on the movable part of the stocking; and this weight must be lifted by each jet of air admitted to the air-lift tube. In view of the great frequency of the air-jets, the enormous amount of useless work which this form of valve necessitates will be apparent. Moreover, the numerous alternate openings and closings of the rubber stocking soon destroy its elasticity and wear it out. The difficulties attending agitation in Pachuca tanks are described by Huntington Adams, in a paper read at the Wilkes-barre meeting of the Institute, and need not be repeated here.[1]

It should also be understood that the efficiency of air-pumping is affected by dimensions of apparatus, etc., differently from that of mechanical pumping. For instance, a mechanical pump

[1] This volume, p. 595.

designed for a 6-in. discharge-pipe will pump as easily the same quantity through a 16-in. discharge. But in the case of air-lift pumping, the volume and pressure of compressed air that would be sufficient to pump violently through a 6-in. discharge-pipe will have no lifting-effect whatever through a 16-in. pipe; for the compressed air would rise in a stream of separate bubbles through the liquid in the lift-pipe, and would not be of sufficient volume to form solid air-disks reaching from wall to wall of that pipe; hence the liquid column would be unbroken and would itself be in hydrostatic balance with that outside the lift-tube, and no displacement would result. This points to the economy of using the smallest air-lift tube consistent with the volume of liquid to be pumped.

The Parral-Tank System of Slime-Agitation.

In this system, designed and developed by me, for which United States and Mexican patents have been obtained, the defects in the Pachuca-tank system above referred to have been eliminated, and corresponding advantages secured.

A complete tank-equipment of this system, consisting of five tanks, and capable of treating 500 tons daily, has been installed at the milling-plant of the Veta Colorado M. & S. Co., at Parral, Mexico. Besides the Parral tanks there are two standard Pachuca tanks, one of which is used as a treatment-tank, and the other for holding the wash-water for the filter-press plant.

The Parral tanks, 25 ft. in diameter and 42 ft. high, are equipped with the special piping and the apparatus peculiar to this system, while one Pachuca tank is equipped with the piping and apparatus of that system. The treatment-tanks (*i. e.*, the one Pachuca and five Parral tanks) have been piped for the individual and continuous systems of treatment, and each of these systems has been tried out, separately, a complete record of the results being carefully kept. No advantage in the extraction of values has been shown by either of these systems over the other; but the continuous system is more economically operated by reason of its great simplicity and "fool-proofness."

Fig. 3 shows the battery of treatment-tanks. On the extreme right is the Pachuca tank, on top of which sits the deck-house

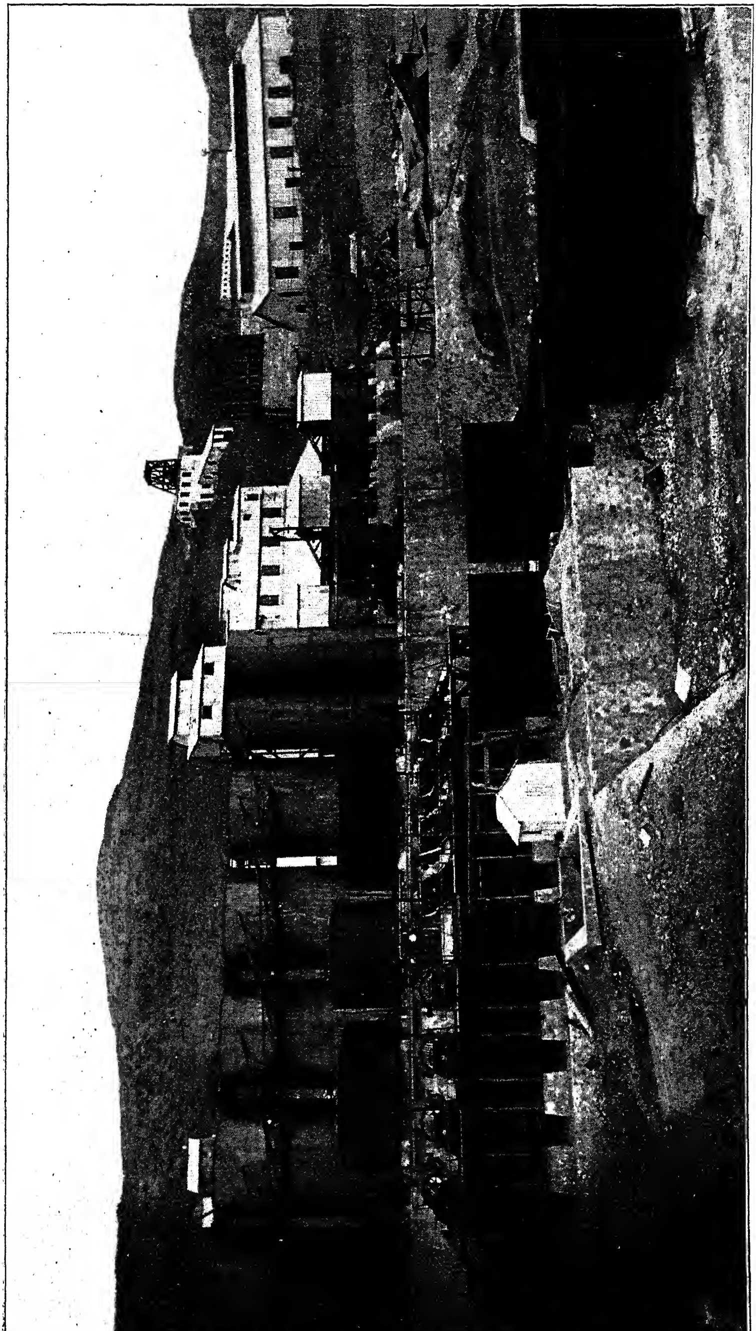


FIG. 3.—SLIME-TREATMENT PLANT OF THE VETA COLORADO M. & S. CO., PARRAL, MEXICO.

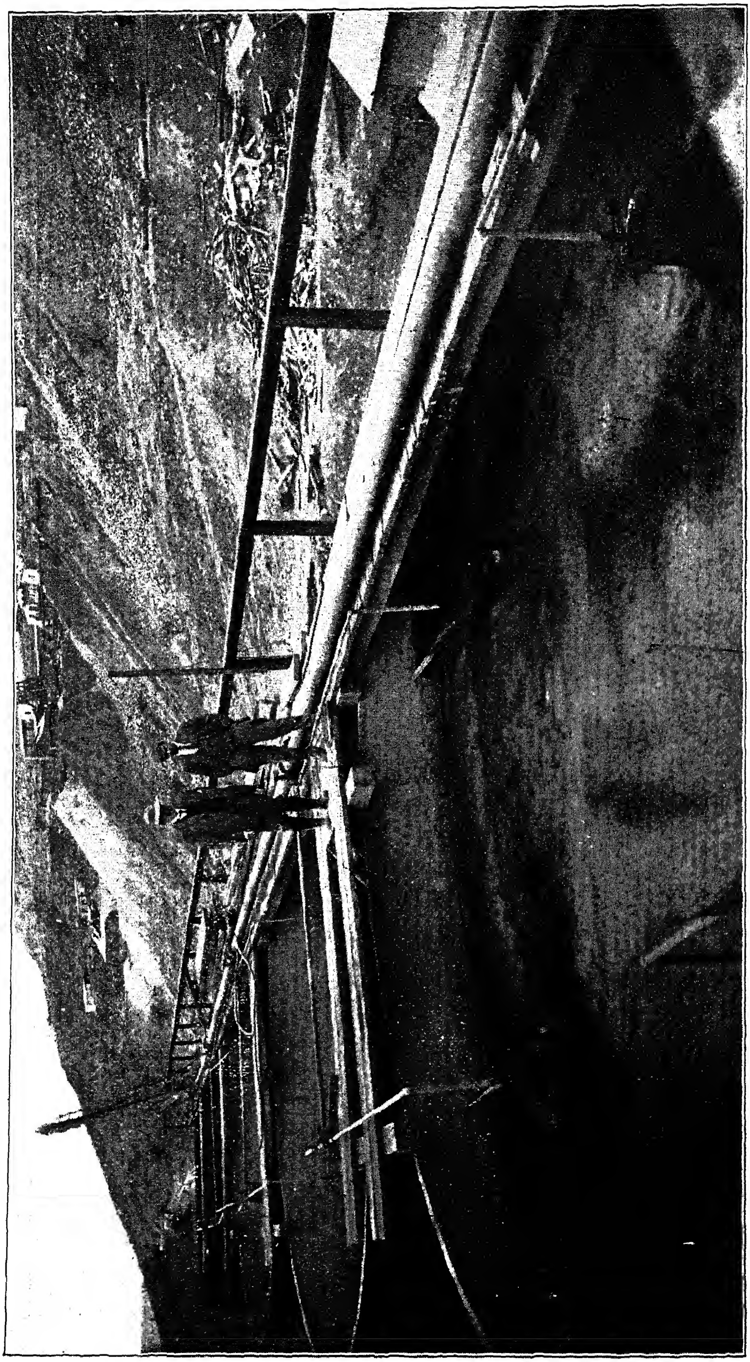


Fig. 4.—View of Treatment-Tanks of Veta Colorado M. & S. Co., Showing Pulp-Discharge from Transfer-Pipes.

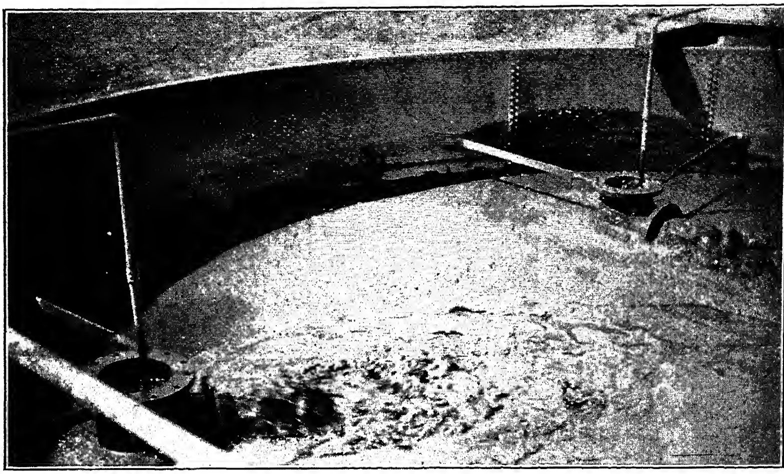


FIG. 5—ROTARY FLOW OF PULP IN PARRAL TANK.

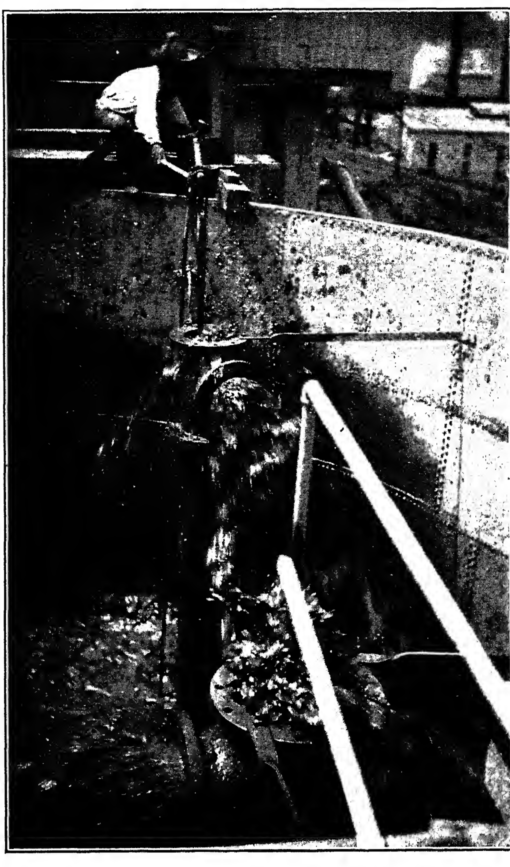


FIG. 8.—COMMENCEMENT OF PULP-TRANSFER IN PARRAL TANK.

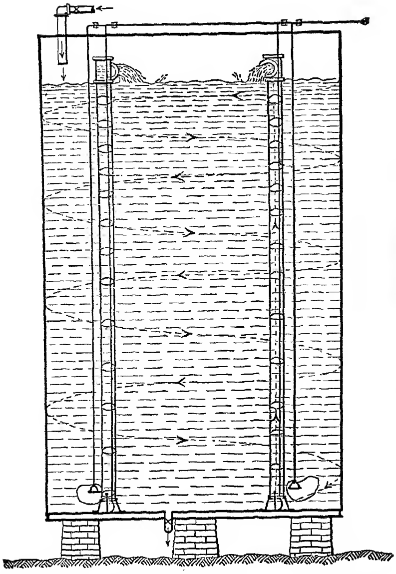


FIG. 6.—PHANTOM-VIEW OF PARRAL TANK, SHOWING ALTERNATE DISKS OF COMPRESSED AIR AND PULP ASCENDING THE TRANSFER-PIPE AND ROTARY TRAVEL OF PULP-PARTICLE.

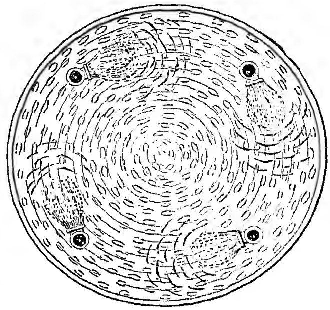


FIG. 7.—TOP-VIEW OF PARRAL TANK, SHOWING ROTARY MOTION SET UP BY DISCHARGES FROM THE TRANSFER-PIPES.

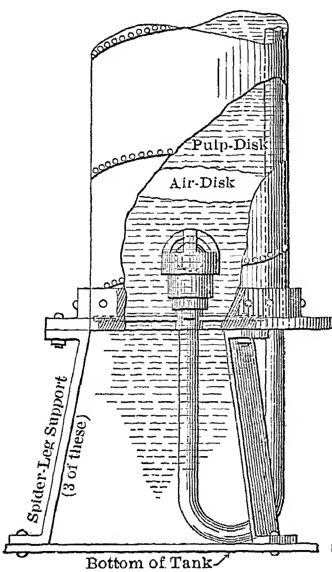


FIG. 9.—COMPRESSED-AIR NOZZLE IN TRANSFER-PIPE OF A

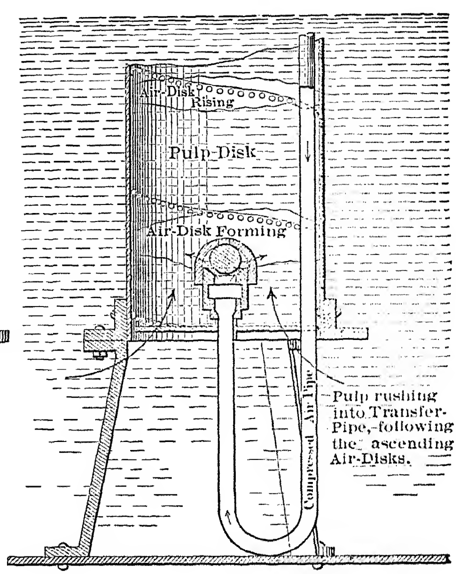


FIG. 10.—SECTION THROUGH TRANSFER-PIPE IN PARRAL TANK.

used for the titration of samples, while in the same row to the left are the five Parral tanks. Along the front of the tanks near their top are seen the piping for the continuous-treatment system, and the sampling-platform. In the center and lower left corner are shown the "excess" tanks and the battery of Kelly filter-presses appurtenant to the plant.

The object of the Parral-tank system of agitation is the same as that already described in reference to the Pachuca tank, but the tank-design and the mechanical equipment used are entirely different from those of the Pachuca system.

The Parral tank is flat-bottomed, 25 ft. in diameter and 42 ft. high, with a capacity three times as great as that of the standard Pachuca tank. For transferring pulp from the bottom to the top of the tank, four 12-in. transfer-pipes are set 12 in. from the bottom, 4 ft. from the tank-side and equi-distant from each other. The compressed air is admitted into these pipes through a patent nozzle fitted with a ball-valve, which automatically opens and closes, intermittently, as required in the jet-feeding of the compressed air. I refer to these as transfer-pipes, this being more accurately expressive than lift-pipes, for, practically speaking, the pulp is not lifted, but transferred from the bottom to the top of the charge.

In case the compressed air should fail, and in the momentary intervals between the jet-issues, the air-nozzle is securely and automatically sealed by the ball falling back on its seat, and the entrance of pulp to the air-pipes is prevented.

On the delivery- or top-ends of the transfer-pipes, tees of equal diameter are bolted, with the run in line with the pipes, and the outlets so directed as to discharge the pulp in line of segment-chords to the circumference of the tank. The discharge of all the transfer-pipes is in the same direction, and the force of the discharge sets up a spiral or rotary flow in the tank-charge which, in a short time, extends down to the bottom of the tank.

Figs. 4 to 7 show the pulp discharging from the transfer-pipes, and the undulations of the rotary flow set up in the tank-charge. The delivery-ends of three of the four transfer-pipes are shown in Fig. 4, but the rotary flow is perhaps more clearly seen in Fig. 5. When Parral tanks are receiving their charge for individual-charge treatment, an auxiliary air-pipe is ex-

tended down alongside each transfer-pipe to a point near the bottom of the tank, and the compressed air issuing from these pipes keeps the pulp in agitation and prevents its settling on the bottom. In the continuous system this pipe is never used.

When the tank is filled to within 10 or 12 ft. of the top, the air is closed off the auxiliary pipes and turned on in the transfer-pipes. Fig. 8 shows a workman making this change and the transfer of the pulp (lift at this time) commencing. This figure shows also the method of making the transfer-pipes fast to the side of the tank, which is very secure and simple.

The spiral flow set up in the tank, as shown in Figs. 6 and 7, carries the pulp-particles round and round, so that the distance traveled by the pulp from the time it is delivered at the top until it reaches the bottom is many times greater than if it settled vertically, as in the Pachuca tank. In other words, the solids are carried in suspension by the rotary flow of the solution as they would be carried in a flowing river; the settlement of the heavier particles is thus retarded; and, consequently, the necessity for transferring the pulp from the bottom of the tank to the top is proportionately lessened, and the cost of the work is comparatively reduced.

In the Parral-tank system no special diameter of tank need be adhered to as in the Pachuca system. The relation of the diameter to the height of the tank may be whatever is economical in holding-capacity, which should be the main consideration in determining tank-diameters.

To secure, under this system, perfect agitation and the necessary rotary flow in tanks of the largest diameter, it would only be necessary to install a proper number of transfer-pipes, with discharge-outlets placed in the right direction to set up and maintain the rotary flow. A Parral tank (see Fig. 2), of the same holding-capacity as a standard Pachuca tank, would be 15 ft. in height by 25 ft. in diameter, and would be equipped with four 8-in. transfer-pipes; while the necessary pressure of compressed air would be only 8.5 lb. per square inch. The comparative cost of construction and operation of these two types of tanks is easily estimated.

TABLE I.—*Comparison of Corresponding Items in Standard Parral and Pachuca Tanks.*

Points of Comparison.	Dimensions or Number. Pachuca.	Parral
Height in feet,	45	15
Diameter in feet,	15	25
Horizontal area in square feet,	176.7	490.8
Effective holding height in feet,	39	14
Holding-capacity in cubic feet,	6,891.3	7,671.2
Holding capacity in metric tons of solids:		
Pulp-ratio: Solution 2, solids 1,	83.3	92.8
Solution 1.5, solids 1,	125.3	139.4
Solution 1, solids 1,	139.5	155.3
Weight of steel plate and all construction-material in pounds,	33,000	14,650
Weight of steel per ton of 2:1 pulp, in pounds,	400	157
Air-pressure required for agitation, in pounds,	30 to 50	8 to 10

The compressed-air nozzle with its ball-valve, which was designed and patented for the Parral-tank system of agitation, may be used in any air-lift, and makes for the highest possible efficiency of compressed air used as a lifting agency. Figs. 9 and 10 illustrate the construction and operation of this valve. An examination of the ball-operation will show that the pressure on it, due to the hydrostatic head of the pulp-charge, is balanced, except for the area of the ball that rests on the seat. The seat-area of the valve, which is 2 in. in diameter, equal to a horizontal area of 3.1416 sq. in., would leave an unbalanced weight of 73 lb. on the ball, if it were to replace the rubber stocking in the Pachuca tank—or 763 lb. in favor of the ball-valve.

As the air-nozzle is called upon to open and close several times each second in permitting the jet-discharge of compressed air into the transfer-pipe, the aggregate of the useless work which the rubber stocking imposes on the compressed air, and the comparative advantage which the ball-valve possesses over it, will be easily estimated. My reason for saying that the probable frequency of the air-jet discharge will amount to several per second, is, that the sounds of the seatings of the ball-valves, as heard by one going underneath the tank, seem almost as frequent as the blows of an air-hammer.

So far, I am not able to fix any period as the useful life of the Parral valve; for these valves have been in operation since

the starting up of the plant, Feb. 6, 1911, and, at a recent date, had shown no signs of wear.

For comparison between the two valves on this point, it may be noted that the Panilla mill contains 12 standard Pachuca tanks, 10 of which were equipped with the rubber-stocking valve of that system, and 2 with the nozzle and ball-valve of the Parral-tank system. These tanks began operation on the first of January of this year; and the rubber stockings soon wore out and were replaced by Parral valves, while the Parral valves originally installed showed, when recently examined, no signs of wear and are apparently as good as ever. In this plant and in that of the Veta Colorado M. & S. Co., the Parral valves never gave any trouble in starting up, even after the air had been closed off for three hours at a time; while, under the same conditions, the valves of the Pachuca tanks were only started after a great amount of trouble.

Although the transfer-pipes in the Parral tanks are 12 in. in diameter, I believe 6-in. pipes would produce sufficient rotary flow in the tank-charge to give the required agitation. In the operation of the tanks installed, when the transfer of the pulp is started and a strong rotary motion (about 10 ft. per second) communicated to the tank-charge, the air-valve is turned down until the flow of pulp from the transfer-pipes is reduced to one-third of their normal capacity, and so continued to the end of the treatment. By repeated tests, it has been shown that the extraction of values was as good with one-third the normal capacity of the transfer-pipes as when they were being operated at full capacity. From these tests it has been deduced that, so long as the spiral flow in the tank is maintained at a speed sufficient to retard materially the vertical settlement of the solids, so as to keep them suspended in proper proportion in the solution, the extraction of gold and silver proceeds just as rapidly as when the pulp is violently agitated.

I have no exact data from which to form an estimate of the comparative amount of air consumed per ton of pulp treated in the two systems, for the air has never been metered; but engineers who operated the valves on the air-pipes of both tanks, experimentally, with a view to estimating the flow of air by the proportional valve-openings, have reached the conclusion that it does not require more air to operate the four 12-in. transfer-

pipes of the Parral tanks than the one 16-in. transfer-pipe of the Pachuca tank; and I venture my personal opinion that when a meter-test of the air-flow is made, this conclusion will be confirmed.

The comparative dimensions of the Parral tanks, as installed at the mill of the Veta Colorado M. & S. Co., and of the standard Pachuca tanks, with the individual equipment of each, are given in Table II. It may be repeated in this connection that 15 ft. is the largest diameter that can be given to the Pachuca tank, while the diameter of the Parral tank may be made as great and the height as low as desirable.

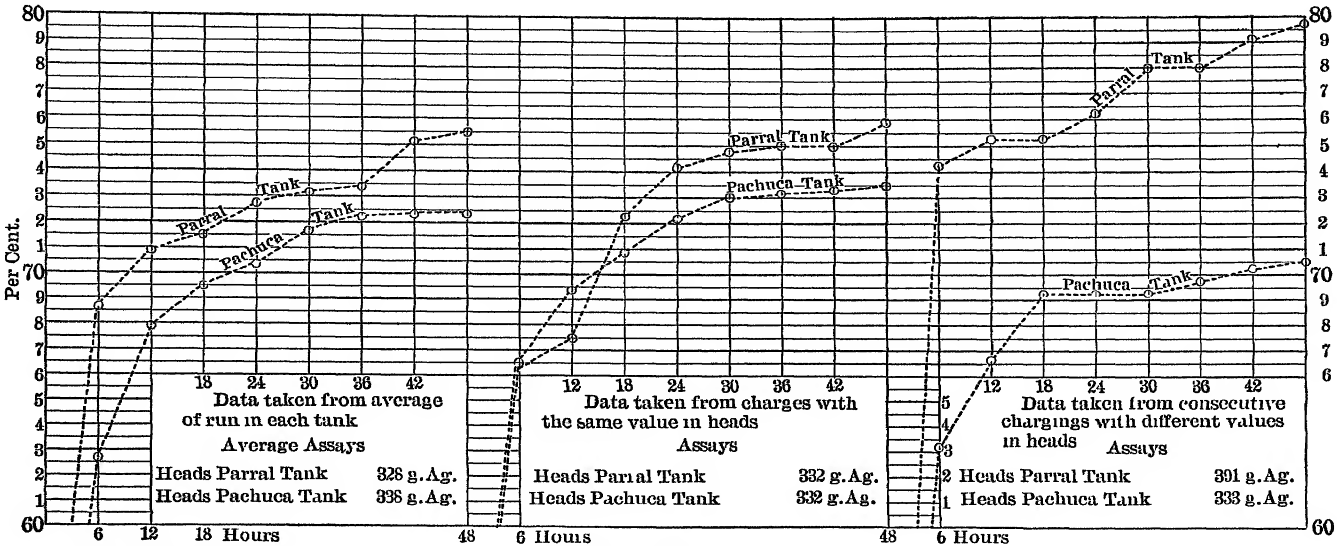
TABLE II.—*Comparative Dimensions of the Parral and Pachuca Tanks and Their Respective Equipment, as Installed at the Mill of the Veta Colorado M. & S. Co.*

Points of Comparison.	Dimensions or Number. Pachuca.	Parral.
Height in feet,	45	42
Diameter in feet,	15	25
Area of bottom of tanks in sq. ft.,	176.7	490.8
Holding-capacity for each foot in height, cu. ft.,	176.7	490.8
Number of air-lift or transfer-pipes,	1	4
Diameter of each air-lift pipe in inches,	16	12
Total cross-sectional area of air-lift pipes, sq. in.,	201	452
Diameter of each compressed-air pipe in lift-pipes in inches,	1.5	2
Total cross-sectional areas of air-pipes in lift-tubes in sq. in.,	1.7671	3.1416
Proportional area of tank-bottom for each sq. in. of cross-section of air-lift tubes, sq. ft,	0.8	1.8
Area of tank-bottom for each sq. in. of compressed-air pipe, sq. ft.,	100	156

This table shows, especially if studied in connection with Table I., that, taking unit against unit in tank-construction and equipment, the Parral tank is the more economical.

Extraction of Values.

An unexpected result became manifest in plotting the time-extraction curves, Fig. 11, from the assay-records of the samples taken, during the treatment-operations, from the Parral and Pachuca tanks when operating on the individual-charge method. The curves show parallel results obtained from the two tanks treating similar pulp under three different conditions.



Chemical Constituents of Solution.		Physical Condition of Pulp.		Extraction
KCN,	15 per cent.	Dilution,	2 to 1	Prior to tank treatment, 10 to 20 per cent.
Protective alkalinity,	850 g.	Specific gravity,	1.26	After treatment, as per chart.
Lead,	3 g.	Screen sizes: + 200, 15 to 20 per cent; — 200, 75 to 80 per cent.		Total extraction, as per chart + 10 to 20 per cent.

Note.—In explanation of the percentage column of the above chart, it may be said that more than 62 per cent. of the head-values were extracted during the first six hours' agitation.

FIG. 11.—EXTRACTION-CURVES PLOTTED FROM RECORDS OF OPERATIONS IN PACHUCA AND PARRAL TANKS, SAMPLES TAKEN EVERY SIX HOURS AND ASSAYED.

Conclusion.

This paper is presented as the announcement of a new and improved system of slime-pulp agitation, for the consideration and criticism of metallurgical engineers connected with or interested in cyanidation. I have given much thought and study to the working out of the design and the development of its mechanical details, and have had the pleasure of seeing my labors rewarded by complete success.

I wish to extend my thanks to William Thompson and Frank Reichmann, the superintendent and engineer of the milling-operations, respectively, who compiled the details of the operations and made the drawings submitted with this paper.

Present Conditions in the California Oil-Fields.

BY MARK L. REQUA, SAN FRANCISCO, CAL.

(San Francisco Meeting, October, 1911)

During the past two years California has developed a new and important oil-field: I refer to Midway. This field produced the famous Lake View gusher, which is credited with a total production in excess of 8,000,000 barrels. Fortunately for the oil industry of the State, this well is now a thing of the past, and nothing save a great crater-like opening marks its location. The pipe is entirely worn away and gone; and it is a matter of serious doubt if there can be anything done that will cause the well to produce again. Fortunately, also, there have been no other wells in that field or elsewhere throughout the State that in any way compared with the Lake View. Midway is noted for large wells, of from 500 to 2,000 barrels production; but the decline is rapid, and a few months serve to bring the output down to a few hundred barrels.

In the oil-territory heretofore blocked out as proved and probable, there have been, during the year, many changes. Some areas which were expected to be fairly productive have apparently failed; others, more strictly "wild-cat," have come in; while in some of the older fields there are properties which are beginning to show evident signs of exhaustion. The total area of proved territory will therefore probably suffer but small

increase, when balances are struck off. The increase of new area has come from extensions of the Midway field, the development of a field in Lost Hills and Belridge, and extensions of the Fullerton-Whittier field in southern California. In these later developments, down to date, the fresh area absolutely proved is not much in excess of 3,000 acres. Recent developments in Coalinga indicate the possible extension of that field to the south, but at great depth. Coalinga is still the most northerly field of any consequence in the State. The Kettleman Hills have hitherto brought in nothing, although a depth exceeding 3,500 ft. has been reached. Much of the territory proved within the year is extremely deep and expensive to develop and operate.

This, however, is not true as regards a narrow strip in the Lost Hills and the proved tract in the Belridge fields, located respectively 26 and 12 miles NW. and N. of McKittrick. In these fields it is claimed that at depths varying from 600 to 1,200 ft., 200- to 500-barrel wells are the rule, producing oil of 23° gravity and higher. So far as can be foreseen at the moment, this territory is the most disturbing factor in the State, as regards the future price of oil. It is yet too early to predict with accuracy the possibilities of these two fields, and especially of the Belridge territory, but that there is oil underlying the locality at comparatively shallow depths, admits of no question. Thickness of sand, saturation, area proved, and sundry other factors necessary to be determined before any estimate can be made, are as yet not obtainable.

Geologically, the ideas as advanced by the U. S. Geological Survey[1] must be altered, at least as regards the areas through the Lost Hills, and in the immediate vicinity thereof.

In the above-cited reports it is declared that the Vaqueros (Lower Miocene) sands become less saturated as they pass southward, and, although their depth below the surface may be calculated in the Kettleman Hills, it is impossible to determine their depth in the Lost Hills with any degree of accuracy. The inference is that the oil will be here found in the Vaqueros (Lower Miocene) sands, as at Coalinga, rather than in the McKittrick (Upper Miocene) beds, as in the productive fields of

[1] *Bulletin No.* 357, *U. S. Geological Survey,* pp. 120 to 124 (1908); and No. 406, pp. 206 to 209 (1910).

the Midway district and other fields to the south, and in smaller quantities.

As is generally understood, the bulk of the oil of the Coalinga field originates in the organic Tejon (Eocene) shales and passes upward into the overlying sands chiefly of the Vaqueros (Lower Miocene) series. In the fields further south, the oil originates in the Middle and early Upper (?) Miocene shales, of similar organic nature, and passes upward to sands of Upper Miocene and Pliocene deposition included in what is known as the McKittrick formation. In the Coalinga field the equivalent of these Miocene shales is probably what is known as the "Big Blue," which is made up of clay, sand, and gravel, but is not organic in nature, and does not therefore possess the essentials necessary to give rise to commercial oil in this vicinity. Passing southward, however, this member increases in organic contents and thickness, and in the Pyramid Hills gives rise to a distinct petroliferous odor on fresh fracture. The thickness has here been estimated at 1,800 feet.[2]

The increase in the petroliferous nature of these Miocene shales as they pass southward, and the fact that they dip under the plain, to be uncomformably covered by McKittrick beds, indicate a possibility of commercial oil in the latter formation, as well as possibly in the Vaqueros sands. That this is an important condition is shown by the actual development of oil in what has proved to be the McKittrick formation in the Lost Hills.

Aside from the developments in the Lost Hills, Belridge, and Fullerton-Whittier districts, there has been nothing of great moment proved, although certain undeveloped localities are recognized as offering possibilities of production at shallow depth.

Naturally, the sudden increase of production caused by developments in Midway has created a large surplus. Consumption has not kept pace with production; and it is highly improbable that consumption will, at any time in the future, increase in any such proportion as in past years. With comparatively few exceptions, home-markets are supplied, and future increase in consumption must come from the increased

[2] *Bulletin No.* 406, *U. S. Geological Survey*, p. 63 (1910).

demands due to larger population and shipments to South America.

If we assume present daily production over a period of eight months ending Sept. 1, 1911, at 211,000 barrels, and surplus at 34,500 barrels, the daily consumption amounts to 176,500 barrels, or 64,422,500 barrels per annum. Compared with 1909, in which year the actual consumption was about 58,000,000 barrels, the increase is not large.

The annual production of oil in California has been as follows:

	Barrels.		Barrels.
1875	3,000	1893	470,179
1876	12,000	1894	705,969
1877	13,000	1895	1,208,482
1878	15,227	1896	1,252,777
1879	13,543	1897	1,903,411
1880	40,552	1898	2,257,207
1881	99,862	1899	2,642,095
1882	128,636	1900	4,324,484
1883	142,857	1901	8,786,330
1884	262,000	1902	13,984,268
1885	325,000	1903	24,382,472
1886	377,145	1904	29,649,434
1887	678,572	1905	33,427,473
1888	690,333	1906	33,098,598
1889	303,220	1907	39,748,375
1890	307,360	1908	48,300,758
1891	232,600	1909	58,191,000
1892	385,049	1910 (estimated)	75,000,000

The field-price at present is approximately 30 cents per barrel for fuel-oil and 45 cents per barrel for refining-oil. There is no real reason why this price should not rule lower, as there are apparently some producers willing and anxious to sell at prices considerably below these figures.

Drilling is still active, although much of the work is being done by the Southern Pacific Co., which is reported to be running over ninety strings of tools. On Jan. 1, 1911, the number of rigs drilling was 567; on July 1, 492. For the six months the total production is approximately 38,000,000 barrels. Consumption has not materially increased for the half year; on the contrary, a falling off has been the tendency for the past 90 days.

To-day there is above ground a total of approximately 40,000,000 barrels. The average surplus for the eight months

ending Aug. 30, 1911, was approximately 32,000 barrels per day. By months the daily average excess has been, commencing with January, 21,000, 30,000, 57,000, 35,000, 18,000, 33,000, and 32,000 barrels.

It is exceedingly to be regretted that the oil-producers of California, as a whole, do not apparently realize the real cost of production. The older fields cannot hope materially to reduce production-costs. On the contrary, as the deeper territory is drilled, and present producing wells decline, costs must inevitably advance. From territory of, say, 2,500 ft. depth, total costs will approximate from 30 to 35 cents per barrel. For direct production—*i.e.*, pumping, cleaning, and pulling—10 cents per barrel may be safely assumed. For maintenance of surface-equipment and rigs, 4 cents is a conservative estimate. For exhaustion of oil-land, and redemption of capital, from 6 to 10 cents must be reckoned; and for drilling to maintain production, 12 cents is not excessive. These figures make a minimum of 32 cents and a maximum of 36 cents. It is obvious that for any business in which the risk is as large as in the drilling of oil-wells, the resultant profit should be in proportion to the risk involved. Under existing conditions in California, this is most emphatically not the case.

The recent agitation which has brought about the dissolution of the Standard Oil Co. has in no way benefited the small producer. On the contrary, the situation has been rendered, if anything, more acute. Because of its self-contained character, as producer, transporter, refiner, and marketer, the Standard Oil Co. was able to earn a profit when the small producer was confronted with a loss. Regulating prices, even within modest limits, by agreement is apparently to-day a criminal act. Because of this, it is not possible to reach any agreement with the great factor in the California oil industry, and we have the spectacle of the Standard Oil Co. of California exerting a stronger and stronger domination, and the small producer getting deeper and deeper into financial difficulty.

The utter failure of "trust-busting," so far as the commercial relief of California oil-producers is concerned, is self-evident. It would be much more to the point if conditions were frankly faced as they exist, and regulation of output and prices permitted, if necessary, under government supervision. What

is being aimed at might be accomplished in that way. It is certainly not being accomplished at present by the absurd methods now pursued. The Standard Oil Co. of California, operating as a strictly local institution purged and purified from contaminating associations with the parent company, can quite as effectively dominate the fields as did ever the parent. And unless we turn anarchists pure and simple, and confiscate property and ignore vested rights, there is absolutely no way of curing the trouble save by pools and agreements recognized and encouraged by law. What is true of the Standard as to the cost of doing business will apply in less degree to the Union Oil Co., and to the Associated Oil Co. in still less degree, because the latter company is not in the refining business. To the small producer, who depends for his profit on taking the oil from the ground and selling it to the transporting and marketing companies, the present conditions spell ruin unless corrected in the near future.

The waste of oil is appalling. Brought to the surface, it is allowed to lie for months in open earthen sumps. Storage-tanks of steel, concrete, and earth are full to overflowing; and yet the daily surplus of from 31,000 to 50,000 barrels accumulates, and is in part dissipated by evaporation. Probably not less than 4,000,000 barrels, and possibly double this amount, of oil was lost last year by evaporation and seepage. This year will see quite as much similarly dissipated. Much of this loss could be eliminated by agreement among the producers. Practical conservation would be along lines of restricted production, permitting the oil to remain in its natural reservoirs underground until such time as it can be produced and sold at prices that will yield a reasonable profit to the small producer. To improve prices and relieve surplus, suggestions have been made that large quantities of oil be burned. This would be an attempt to conserve prices at the expense of natural resources. The mere suggestion of such a remedy for a condition that need not exist if sane conservation were effective, is sufficient commentary on the utter inability and ineffectiveness of theoretical cures. Thanks to existing laws, it seems that we must continue recklessly to squander our resources and rob the State of one of its greatest assets without satisfactory return.

On the Pacific coast of North and South America there has as yet been developed no deposit of coal equal in quality to the best eastern Australian or Welsh products. The cost of the non-uniform article which is found and mined in Washington and British Columbia is much higher, as must also be similar products awaiting development in Peru and Alaska. Excess in these coal-costs and the poor quality of the article have, heretofore, not only retarded various industrial developments, but hindered manufacturing enterprises on the Pacific Coast. This condition, however, paved the way for the introduction, eager use, and marked success of the fuel par excellence in steam-generation—California oil.

A few comparative statements showing its superiority to coal in point of heat-value and economy in firing boilers follow:

California oil in general use and under identical conditions gives uniform results. The evaporative power of the Pacific Coast coals varies greatly. Under horizontal boilers, 1 lb. of California oil should evaporate from 13 to 15 lb. of water. One pound of the best coal in use on the Pacific Coast will hardly evaporate 9 lb. of water, and 6 lb. is the figure for poorer grades. Taking the ratio of the two fuels in point of evaporation efficiency as 14 lb. to 8 lb., or 1.75 to 1, we find that 1,280 lb., or 3.8 barrels, of fuel-oil is equivalent to one long ton, or 2,240 lb., of coal. In transportation-cost, the advantage in favor of pipe-line is so great that the cheapest rail-transportation cannot compete, although water-shipments come nearer to so doing. Loading- and unloading-costs, losses from wastage and theft, and the difference in stoking-expenses are to a high degree in favor of the liquid fuel.

"Probably no more striking way of actually showing the relative commercial value of coal and oil as a fuel, could be presented than by stating that the Atchison, Topeka and Santa Fé Railroad Company made the following comparative tests, of the cost per train mile, of coal costing \$6.65 per ton and petroleum costing \$1.33 per barrel.

"Twenty-five passenger and freight engines on a thirty-day run, used 2,077 tons of oil and traveled 87,063 miles, or 41.9 miles per ton, or 3,500 miles per month per engine. Oil at \$1.33 per barrel would, at this figure, cost 14.4 cents per mile. Twenty-five passenger and freight engines (same days, same track, and same condition) burning coal, cost 23.2 cents per mile. The oil was 15° Baumé, about the same as the Kern River oil, which is 14° and 17° Baumé; this showed a saving for oil of 38 per cent., and the experiment was tried with coal at \$6.65 per ton.

"In this extended and practical test the cost of the oil per barrel was one-fifth of the cost of the coal per ton, while the resulting gain for oil was 38 per cent. Stated in another form, the value of the two fuels would be the same when the price of the coal in tons was three and one-half times the price of the oil in barrels "[3]

The following tables, extracted from a report compiled at my request by George W. Dickie, consulting marine engineer, of San Francisco, will be of interest in practically illustrating the proposition. Oil is figured at $1 per barrel. Indicated horse-power of steamer, 3,000; steaming speed, 11 knots.

"A."

Cost of Oil Fuel Per Day.	Quality of Coal in B t u. Per Pound.	Tons Coal Per Day	Added for Additional Labor.	Cost Per Day for Several Qualities of Coal at the Following Prices, Delivered		
				$4.	$6.	$8
	12,000	62.70	37.56	$288.36	$413.76	$513.16
$300	11,000	68.40	37.56	311.16	447.96	584.76
	10,000	75.20	37.56	338.36	488.75	639.16
	9,000	83.60	37.56	371.96	539.16	706.36

"B."

A vessel engaged in coastwise traffic between California ports:

Oil-consumption per trip, 4,000 barrels,	$4,000
Firemen, wages and food,	275
Total cost,	$4,275

Coal-consumption per trip.

1,200 tons, at say $4,	$4,800
Firemen, wages and food,	1,000
Total,	$5,800
Saving per trip in favor of oil,	$1,525
Assuming two voyages per month, the saving is,	3,050
Allowing 11 months' operation per year, yearly saving,	33,500
Or, 6 per cent. on a sum slightly under,	560,000

This figure of $1 per barrel at San Francisco bay would equal about 65 cents net to the producer at the well.

The United States Geological Survey has estimated the contents of the probable oil-lands in the United States as follows:

[3] *Report of U. S. Naval "Liquid Fuel" Board*, Bureau of Steam Engineering, U. S. Navy Department, pp. 390 to 391 (1904).

Estimated Quantity of Oil in United States.

	Minimum Barrels	Maximum. Barrels
Appalachian field,	2,000,000,000	5,000,000,000
Lima-Indiana field,	1,000,000,000	3,000,000,000
Illinois field,	350,000,000	1,000,000,000
Mid-Continent field,	400,000,000	1,000,000,000
Gulf field,	250,000,000	1,000,000,000
California field,	5,000,000,000	8,500,000,000
Minor fields,	1,000,000,000	5,000,000,000
Total,	10,000,000,000	24,500,000,000

In other words, of the minimum of 10,000,000,000 barrels, California is credited with one-half of the entire possible production of the United States, and of the possible maximum, California may possibly produce one-third.

Personally, I believe that the maximum will unquestionably be in excess of 8,500,000,000 barrels for California. The total production for the State to Sept. 1 was approximately 434,000,000 barrels, leaving a very large percentage still underground. It is safe to say that California oil will dominate the fuel-market on the Pacific Coast during the present century and probably far into the next century. Unless consumption is tremendously increased, this is undoubtedly true. These figures are, of course, only relative approximations, but are sufficiently accurate to warrant the assertion that California oil will dominate the fuel-market of the Pacific at least through the present century.

Comparing California oil with Alaska coal, it is apparent that oil has complete control of the field.

Alaska coal can be landed at Puget sound ports for approximately $4 per ton.[4]

Assuming 3.5 barrels of oil as equal to one ton of coal and oil at 50 cents per barrel at the well, its comparative cost with coal per ton delivered on Puget sound would be $3.50, and with oil at 75 cents at the well, this cost should not exceed $4.20. At prices even in excess of this, consumers would not return to coal, owing to the many indirect advantages accruing to the burning of oil. Costs at other points depend entirely upon distance by sea. Assuming Valparaiso, Chile, as the southern, and Douglas Island, Alaska, as the northern ex-

[4] *Bulletin No.* 442, *U. S. Geological Survey*, p. 88 (1910).

treme, with oil at 60 cents per barrel at the well, coal must sell at $5 per ton at Valparaiso, and $3.50 at Douglas Island, in order to equal oil in fuel-value. This takes into consideration due allowance for interest, redemption-funds, depreciation, and transportation. When the prices of oil are yet higher coal cannot compete, because the oil is so much more satisfactory in every way, and has so many advantages, that the cost of coal would have to be materially less to induce the abandonment of oil. In view of the above statements, it is fair to assume that during the life of the fields there will be no fear of competition from coal until oil is selling above 75 cents per barrel.

Recent experiments indicate the possibility of oil being used for domestic purposes, even in small dwellings. I am using it in my home for both cooking and heating, to the entire exclusion of coal; and a more recent device seems to make the installation-cost so small as to open the entire domestic field to oil-competition. If so, the consumption of coal will practically cease in California, and the public will cut its fuel-bills more than 50 per cent.

The action of the government in withdrawing certain territory is a step in the right direction. Additional drilling at this time would benefit no one, and would be an additional menace to an already overburdened situation. There is no storage so satisfactory as that afforded by the underground reservoirs from which the oil comes. It is free from costs of any kind, and seepage and evaporation are entirely eliminated. Some plan, however, should be decided upon, whereby the land will be available when needed. Leasing under certain restrictions would seem to be a logical solution. At present it would be folly to open in any way this withdrawn area. Territory now producing can care for consumption for an indefinite period. As a suggestion, I should say that government land should not be leased so long as oil at the well sells for less than from 60 to 70 cents per barrel, and that, on leases so granted, no new drilling should be permitted when prices rule below this figure. This would be sane and practical conservation, as it would permit production only in times of need, and would conserve a great natural resource that, once exhausted, can never be replaced.

Gold-Production in California.

BY CHARLES G. YALE,* SAN FRANCISCO, CAL.

(San Francisco Meeting, October, 1911.)

A FEW years ago somebody connected with one of those self-constituted bodies of unofficial character, like a Chamber of Commerce, Board of Trade, or State Development Board, started a catch-phrase referring to California as "The Land of Sunshine, Fruit, and Flowers," and the railroad magazines and folders keep it steadily in use, working day and night. Yet it altogether ignores the substance which brought the State into the Union, which peopled it, and which made it famous throughout the world. You ladies and gentlemen who have come from what we here call "the East," have in your own States, no matter which one, sunshine, fruit, and flowers. But your Eastern States, having these things as we do, have not the gold that we do. Therefore, the old designation of "The Golden State," applied to California, should be revived, as being the most distinctive term. It is worthy of remembrance, too, that during the dark days of the civil war this State handed over $172,000,000 in yellow gold, and saved the credit of the nation.

Gold-mining has been carried on in California since

"The days of old,
The days of gold,
The days of '49,"

and it still continues. Since that historic year, and up to the end of 1910, the State has produced, in gold alone, $1,530,214,468. Since 1792 the entire United States production of gold has been $3,261,573,500, so that the single State of California has, in that period, produced within $201,144,564 of one-half of all the gold from Alaska, Arizona, Colorado, Idaho, Montana, Nevada, New Mexico, Oregon, South Dakota, Utah, Washington, and the Southern and scattering States. In other words, all the other 25 gold-producing States of the United States combined

* Statistician of the U. S. Geological Survey.

have only produced about two hundred millions more than the single State of California has in the long period of 118 years. Moreover, it has taken California but 62 years to produce that near half, which it has done at the average rate of $24,680,878 per annum. This shows an average gold-yield of $2,056,739 per month for the last 62 years.

California therefore deserves the title of "The Golden State."

It is to be noted, moreover, that California is still the leading gold-producer among the States of the Union, and there are still a larger number of producing gold-mines here than in any other State. Gold is being mined in larger or smaller quantities in 34 of the counties of the State.

Among other mining States of the Union, California has, as a gold-producing region, the distinction of holding the record on all counts. It has made by far the largest aggregate product; made the largest output in any single year; made the highest annual average, although its mines have been worked for more than 62 years; kept the lead as a gold-producer the greatest consecutive number of years; has the largest number of individual gold-mines; pursues the greatest number of varied branches of gold-mining; and has the widest geographical distribution of its gold-deposits.

The gold-belt of the State extends its extreme length from Oregon on the north to Arizona and Mexico on the south. Gold is mined in the highest parts of the Sierra Nevada mountains, the foot-hills, the valleys, and on the beaches bordering the ocean. The gold is taken from quartz, placers, pockets, seam-diggings, hydraulic drift, ocean-beach sand, by dredging, wing-damming, dry-washing, and other forms of mining. The snowy ranges, the river-beds, the beaches, the desert sands, the ancient buried rivers, the superficial gravel-deposits, all yield their quota. The climatic conditions in all except the higher ranges are favorable to work the year round. In some of the foot-hill counties, the men work their orange- or olive-orchards and vineyards in the summer and drift for gold under them in the winter months. It is to be noted that to-day the three great dredging-fields are at points where citrus fruits first ripen. The county producing the most gold is in the valley, below the foot-hills, and not in the snowy mountains.

It is not my intention to read you a statistical paper or bore you with a lot of figures, but rather to convey an idea of the present condition of the gold-mining industry in the State as far as it may be done briefly. A few figures are, however, necessary. It may be said that the record year of gold-production in California was 1852, when the placer-miners produced gold to the value of $81,294,700. In 1883 the yield was $24,316,873, and then the annual product gradually declined, owing largely to the closing of hydraulic mines, until, in 1889, the output was only $11,212,413. For seven of those years, between 1883 and 1904, it was less than $13,000,000 annually. Since 1904, the gold-yield has averaged about $19,000,000, sometimes exceeding $20,000,000, and it is to be confessed there is not much prospect of an increase. With labor at $3 per day, and an 8-hr. day enforced by law, it is difficult for the quartz-miners to make much profit on ore of ordinary grade unless large ore-bodies are worked, and as a consequence many have been compelled to cease operations. Still, the tonnage from the deep mines continues to be of considerable proportions, this having been 2,697,885 tons last year, of which 1,963,296 tons were siliceous or gold-ores. The average value in gold of this ore was $5.20 per ton. In some counties, where the veins are comparatively small, the values run up to $8 per ton. Taking a typical large mine in one of these counties, where nearly 100,000 tons were milled, the average yield per ton was $13.68, and the profit $7.51 per ton, over all costs of operation and development.

In the Mother Lode counties, where the ore-bodies are very wide, the ore is low grade. In one of these counties last year, 547,873 tons of ore were milled, yielding an average of $4.69 per ton. But taking all five of the Mother Lode counties, where 1,170,497 tons were milled, the average yield per ton was only $3.78.

It may be a surprise to some to know that, contrary to general supposition, the placer-mines of the State are now yielding 45.09 per cent. and the deep or quartz-mines producing 54.91 per cent. of the entire gold-product. About this proportion has prevailed for several years. It is true that the ordinary surface-placers, where they use rocker, tom, and sluice, now cut but a small figure, but the drift- and hydraulic mines are still

yielding, and the dredgers are now producing 84.94 per cent. of all the placer-gold. This comparatively new system of surface-mining has given renewed life to placer-work. Owing to adverse legislation, the hydraulic mines, formerly highly productive, are now yielding only 7.15 per cent., the drift-mines 5.82 per cent., and the surface- or sluicing-mines only 2.09 per cent. of the placer-gold. Since 1899 the dredges have dug out $40,318,775, and are now producing at the rate of $7,550,000 per annum, with 71 machines in operation. The details of this dredging-work will be given in the paper, Present-Day Problems in California Gold-Dredging, by Mr. Janin, presented at this meeting.[1]

The largest production of gold in California in 1910 came from Yuba county, mainly from dredging. The county most productive in gold from deep mines is Amador, one of the Mother Lode counties. The leading hydraulic-mining county is Trinity, and the leading drift-mining county is Placer. The largest production from dredge-mines was from Yuba county.

It is to be confessed that little progress is being shown in the deep mining for gold, or even in the placer fields, aside from dredging-operations. Even in the latter, in the Oroville field, a decrease in gold-output is already apparent, owing to some of the ground having been worked out, but the increased output of the Yuba field, and in outside districts, made up for the loss in the Oroville dredging-field. There are only three large dredging-fields in the State, these being at points where the Feather, American, and Yuba rivers leave the foot-hills to enter the valley-lands, after, in their course, having cut through beds of auriferous gravel and depositing the fine gold with the soil carried down, when the streams are suddenly arrested from their swift flow by reaching level ground. There are numerous isolated points, however, in other counties, where the circumstances permit the operation of one or more dredges within restricted areas. For this reason dredging is being carried on in 10 counties of the State.

The speculative era of gold-mining has almost entirely disappeared from California. The stock of no single gold-mine is listed on the Stock and Exchange Boards or publicly dealt in. The mining-work is now almost entirely carried on by organized

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[1] This volume, p. 855.

companies which provide capital for the enterprise. The day of the nomadic miner is virtually at an end, and the men are now nearly all employed at daily wages. Of course, there are still many prospectors, but most of the miners live in permanent thriving towns near the larger properties, far different from the old-fashioned primitive mining-camp. High-priced officials have been replaced, office-force and expensės reduced, and only skilled men employed. More railroads, better wagon-roads, cheaper supplies, improved methods of transportation, better machinery at lower cost, highly improved reduction-methods and appliances, adoption of proved modern processes, careful saving of concentrates, stronger powder, power-drills, electric and water-power, heavier and larger milling-plants, more extensive development, and generally improved systems and appliances, have all contributed in recent years towards a change for the better in gold-mining in California.

Examination of Dredging-Properties.

BY FRANCIS J. DENNIS, SAN FRANCISCO, CAL.

(San Francisco Meeting, October, 1911.)

MANY factors govern the value of dredging-ground, and much capital can be wasted by the mistaken policy of contracting for the purchase of property and the installation of machinery before a thorough examination has been made. To the uninitiated investor the presence of gold is generally the criterion, and very superficial evidence is necessary to satisfy him as to this point. He considers the comprehensive report of a competent engineer as a wasteful extravagance, and cannot understand why the engineer requires so much time and money to ascertain the information on which to base his conclusions, when the promoter can furnish him such pleasing and satisfactory data with but little expenditure of time and money. The uninitiated investor will often optimistically risk capital for purchase and equipment, and not until the venture comes to grief does he learn that the conditions are wholly unsuited for dredging. In many instances, a short preliminary examination by a competent engineer would have disclosed these facts. The mere presence of gold is by no means sufficient; some of

the other factors necessary to be ascertained in determining the value of placer-ground for dredging-purposes are: (1) character and distribution of gold, and how much can economically be recovered; (2) character of bedding underlying the gravel, its contour, and whether its gold can be recovered by ordinary dredging-operations; (3) area and depths of gravel, surface contour, over-burden, water-level, proportion of fine material and boulders, and the presence of any material which might interfere with the dredging-operations and the recovery of the gold; (4) water-supply, power available, labor, transportation, and supplies, and cost of these; (5) climatic conditions; (6) title to property, cost and royalties, and legal obstacles to carrying on dredging-operations.

A brief reconnaissance may be sufficient to determine that some of the essential conditions for successful dredging are lacking and no further expense need be incurred. Exposures in the gullies, pits, and shafts often afford considerable evidence of the extent and characteristics of the gravel and the contour and character of the bedding, and this may be readily supplemented by sinking a few additional shafts or drilling a few holes. The preliminary report proving satisfactory, arrangements should be made for thoroughly testing the ground. The area should be surveyed, and the shafts or drill-holes placed according to the sampling-scheme adopted. Information obtained during the preliminary examination should be of use in forming this plan. Where the deposit of gravel and occurrence of gold are fairly regular throughout the area, it is generally laid out into squares of from 200 to 500 ft. Where the occurrence is in channels, this method cannot be pursued, and more judgment and ingenuity are called for in placing the holes and in making estimates from the results obtained. Holes may be placed at short intervals across the channels in rows at regular intervals, and it is sometimes the practice to arrange the holes so that those of alternate transverse lines form longitudinal lines. Whether shafts shall be sunk or holes drilled is a question of expediency. Shafts afford the most complete information, and where round shafts can be cheaply sunk, this method is advisable. Where expensively timbered shafts are required and water and other features militate against shaft-sinking, the cost is prohibitive and drilling is resorted to. It is by this

latter method that most of the dredging-ground in California has been prospected, but it must be borne in mind that the prospectors previously had considerable information as to the characteristics of the gravel and bedding.

No. 3 Keystone drills are usually used in making drilling-tests, but as the plant is heavy and somewhat difficult to transport, the development of the hand-drill in recent years has made it an important factor in the prospecting of gravel-areas in foreign countries, or in localities difficult of access where the transportation-charges are high. Its low first-cost—about one-half of that of a steam-drill—the great reduction in weight—about one-tenth of a non-traction or one-fifteenth of a traction-drill, exclusive of supplies—and the further fact that the whole hand-drill outfit, weighing a little more than 1,000 lb., can be made up into packs, with a maximum weight of less than 75 lb. each, is of considerable importance in prospecting. A complete and very interesting article on the Empire hand-drill for prospecting-work has been published by J. Power Hutchins and Norman Stines.[1] In new territory it is advisable, even though expensive, to sink a few shafts in the initial stages of the investigation, and the data thus obtained will enable a closer interpretation of the character of the ground passed through in drilling.

It is not intended that this present paper shall discuss the details of drilling- and sampling-operations, types of drills, and methods of determining the gold-content of samples. But it should be borne in mind that reliable, experienced men should be employed in this work, and that constant vigilance should be exercised. Field- and time-books should be conscientiously kept, assay-values at various depths noted, characteristics of the bedding and gravel, time consumed, difficulties encountered, and features that might militate against dredging recorded. This information should be kept in the prospecting log-book, which at the finish should contain a summary of all data obtained during the progress of the drilling. From this the engineer should then allocate the results to the proper area, eliminate unprofitable areas where practicable, and summarize the yardage and value of the area that should be worked. In

[1] *Mining and Scientific Press*, vol. cii., Nos. 1 and 4, pp. 39 and 164 (Jan. 7 and 28, 1911).

making this estimate there is no fixed formula for discounting the results indicated by the drilling-test, but experience has shown that the amount of gold obtained by dredging is generally only from 75 to 80 per cent. of that indicated by prospecting. Having ascertained that the conditions are favorable for dredging, it is then incumbent on the engineer to determine on the type and size of dredges, the number to be installed, and the general campaign to be followed.

Having a given area, the yardage and contents of which can be estimated with considerable certainty, he is called upon to decide what equipment will yield the best economic results. His information of the physical characteristics of the area, together with his general knowledge of what is being accomplished in other fields, should enable him to estimate closely operating-costs with dredges of various capacities and construction. A small yardage will evidently not justify a large and expensively-constructed dredge, nor would the extra expense of construction necessary in a dredge for heavy ground be justified in constructing a dredge to work a similar yardage of lighter ground. Amortization of the cost of equipment should be set off against operating-costs, and no extra expenditure be incurred that will not be justified by a corresponding reduction in operating-costs. For example, assume an area of 100 acres of gravel, 11 yd. deep, and containing 5,000,000 cu. yd. A 13.5-ft. boat would cost about $250,000 and would work out the area in less than two years. Assume the operating-cost to be 4 cents per yard, the amortization of the equipment would be 5 cents per yard, making a total of 9 cents per yard. A 5-ft. boat would cost about $85,000 and would work out the area in about five years. Assume the operating-cost to be 6 cents, the amortization of the equipment would be 1.7 cents, a total of 7.7 cents per cubic yard. The installation of the smaller capacity boat would be clearly advisable. Assume the acreage to be 300 and the yardage 15,000,000. The operating-cost of the large boat would be 4 cents and the amortization 1.666 cents, a total of 5.666 cents per yard. Two 5-ft. boats at a total cost of $170,000 would be required to handle this yardage. The operating-costs would be 6 cents and the amortization 1.133 cents, a total of 7.133 cents per cubic yard. Here the installation of the larger boat is clearly advisable. It is

not thought necessary to enter into the refinements of interest calculations in the above examples, but where the operating-profits are large, this factor is well worthy of consideration.

The value of the plant as an asset after the area has been exhausted should also be taken into consideration. The engineer is, of course, presumed to have taken cognizance of, and provided for, the amortization of the initial investment in presenting his report to his principals. Moreover, he may have been sent to report on the area as a dredging undertaking, and, although he finds that it is not suitable for simple dredging alone, some modified form or some method which is an outgrowth of the industry may be profitably undertaken. It is incumbent upon the engineer to use the greatest care in ascertaining information about the property, and the greater his ability and experience the more valuable will his report be to his principals.

Present-Day Problems in California Gold-Dredging.

BY CHARLES JANIN, SAN FRANCISCO, CAL.

(San Francisco Meeting, October, 1911.)

THE first successful bucket-elevator dredge to operate in California was put in commission at Oroville in March, 1898. There had been numerous previous attempts at dredging, but none of the earlier boats proved a success. The gold-miners in California early conceived the idea of a machine to dig gravel from the beds and bars of auriferous streams that were inaccessible by the methods then employed, and it was only a few months after the discovery of gold in California that such a machine was shipped around Cape Horn from New York to San Francisco. This was, however, but the forerunner of many failures in gold-dredging, and was soon at the bottom of the Sacramento river. During succeeding years many other unsuccessful attempts were made, and it was not until 1897 that a dredge of the single-lift bucket-elevator type was floated on the Yuba river. This dredge was built by the Risdon Iron Works for R. H. Postlethwaite, and would probably have been a success if it had been operated on some of the rich Oroville ground

instead of in a turbulent stream, where the dredge was wrecked during a flood, and was not repaired.

Fig. 1 is a sketch-map of California, showing gold-dredging areas.

It is not my intention to narrate in detail the history of the early failures in gold-dredging, and the various steps in the development of the modern boat, but merely to touch upon this in a general way, and to call attention to the wide difference in capacity and operating-cost between the first successful dredge, with an actual capacity of 600 cu. yd. per day—though its rated capacity was in excess of this—and the present modern dredge with 15-cu. ft. buckets, and an average capacity of 250,000 cu. yd. per month. Even this enormous capacity has several times been exceeded on monthly runs. The first successful dredges in California were equipped with open-connected buckets, were operated on head-lines, and had short-tray tailings-stackers. For a number of years dredges of this type were used with varying success, generally on shallow and easily-dug gravel. When attempts were made to work deeper ground and cemented gravel had to be handled, it was found that these first boats were too light, and it was necessary to install heavier machinery to withstand the increased strain.

The modern California-type dredge, with close-connected buckets, spuds, and belt-conveyor for stacking tailings, was a gradual development through years of experimenting. This dredge embodies the ideas of successful operators, and it is generally conceded that dredge-construction and operating-methods in California are far ahead of those in any other country in the world. The dredges built in California cost from $25,000 to $265,000 each; a standard 8.5-cu. ft. boat costing from $150,000 to $175,000, according to conditions to be met in operation. With great improvements made in dredge-construction, and corresponding reduction in operating-costs, areas that were at first considered too low-grade to be equipped with a dredge are being profitably worked, and the gold-production from this source, according to the U. S. Geological Survey reports, increased from $18,847 in 1898 to $7,550,254 in 1910, being 28.3 per cent. of the total gold-production of the State from all sources for the last year, and 84.9 per cent. of the total placer-gold for the year. The production by dredging during

1911 is estimated, as closely as can be figured at this date, at $8,000,000. Table I. shows the production by years of gold won from dredging-operations in California from 1898 to 1911, being a total of more than $40,000,000.

TABLE I.—*Production by Dredges of Gold in California, Years 1898 to 1910.*

Year	Amount	Year.	Amount.
1898,	$18,847	1905,	$3,276,141
1899,	133,812	1906,	5,098,354
1900,	200,369	1907,	5,065,437
1901,	471,934	1908,	6,536,089
1902,	801,295	1909,	7,382,950
1903,	1,488,556	1910,	7,550,254
1904,	2,187,038	1911,[a]	8,000,000

[a] Estimated.

California dredges vary in size from 3.5- to 15-cu. ft. buckets. In Alaska some dredges are equipped with buckets as small as 1.25 cu. ft. to dig shallow ground, and are reported to be working profitably. A 15-ft. Marion dredge has recently been installed on the Boyle concession in Yukon Territory. The successful operation of this boat will no doubt encourage and be followed by further installations of the larger-sized boats where conditions warrant in the Far North. While electricity is the ideal power for operating dredges, steam has been successfully used on a number of installations, and experience has proved the merits of the gasoline- and distillate-engine for this work. There seems little doubt but that the successful development of the gas-producer for the generating of electric power will prove an important factor in considering future dredging of gravel-areas in districts where electric power or water-power for the installation of hydro-electric plants is not at present available. While it is unnecessary to go into the details of dredge-construction in this article, a short description of one of the modern dredges may be profitably given here. A fuller description of a dredge of this character has been published,[1] also a complete record of dredges constructed in California,[2] written by W. B. Winston and Charles Janin.

Yuba No. 13, one of the largest gold-dredges operating in

[1] *Mining and Scientific Press*, vol. ciii, No. 15, p. 446 (Oct. 7, 1911).

[2] Gold Dredging in California, *Bulletin No.* 57, *California State Mining Bureau.*

California, was put in commission at Hammonton, in Yuba River basin, Aug. 10, 1911. This dredge, Fig. 2, was built by the Yuba Construction Co., and is one of five practically similar dredges built by the same company this year. It required 820,000 ft. of lumber for the hull and housing the hull; its dimensions are 150 by 58.5 by 12.5 ft., with an overhang of 5 ft. on each side, making 68.5 ft. total width of housing. The digging-ladder is of plate-girder construction and designed to dig 65 ft. below water-level, and is equipped with ninety 15-cu. ft. buckets arranged in a close-connected line. The entire weight of the digging-ladder and bucket-line is approximately 700,000 lb. The washing-screen is of the revolving type, roller-driven, and is 9 ft. in diameter by 50.5 ft. long and weighs 111,721 lb. Two steel spuds are used, each weighing over 44 tons. The ladder-hoist winch has a double drum, and weighs 67,016 lb. The swinging-winch consists of eight drums, and weighs 34,193 lb. The stacker-hoist winch weighs 3,732 lb. The gold-saving tables are of the double-bank type and have an approximate riffle-area of 8,000 sq. ft. The tailings-sluices at the stern can be arranged to discharge the sand from the tables either close to the dredge or at some distance behind. The conveyor stacker-belt is 42 in. wide and 275 ft. long, on a stacker-ladder of the lattice-girder type, 142 ft. long. Nine motors are in use on the dredge, with a total rated capacity of 1,072 h-p. The total weight of hull and equipment is 4,640,862 pounds.

Natoma No. 10 dredge, now under construction, is equipped with 15-cu. ft. buckets, and will have a steel hull, being the first dredge operating on a steel hull in California. The hull will be 150 by 56 by 10.5 ft. and will have a total weight of 920,000 lb. This is about one-half the weight of a wooden hull to carry the same machinery, and the draft of the boat will be considerably lighter. This boat will be in operation in April, 1912.

Owing to the financial success of gold-dredging, most of the gravel-areas of California have been explored. It is hardly to be expected that any new fields as rich as those now being worked will be found, but it is possible that areas considered unprofitable for dredging, even within recent years, will be worked in the future.

Table II. gives in a general way the approximate extent of dredging-ground in the best-known dredging-districts in Cali-

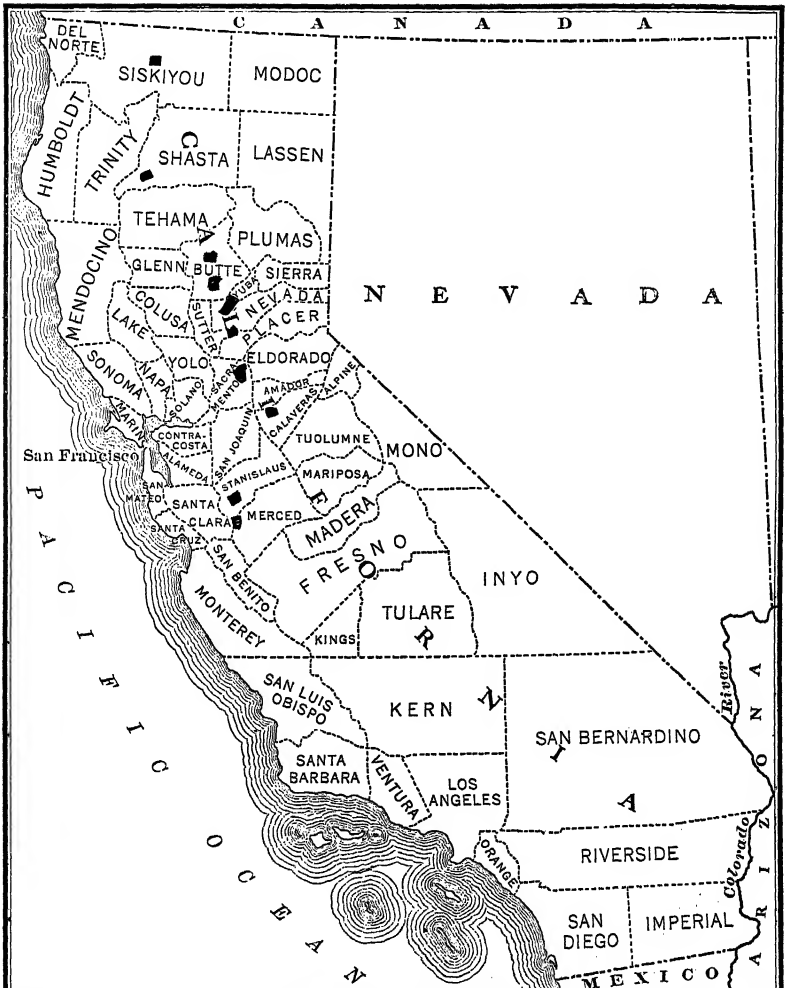


FIG. 1.—SKETCH-MAP OF CALIFORNIA, SHOWING GOLD-DREDGING AREAS.

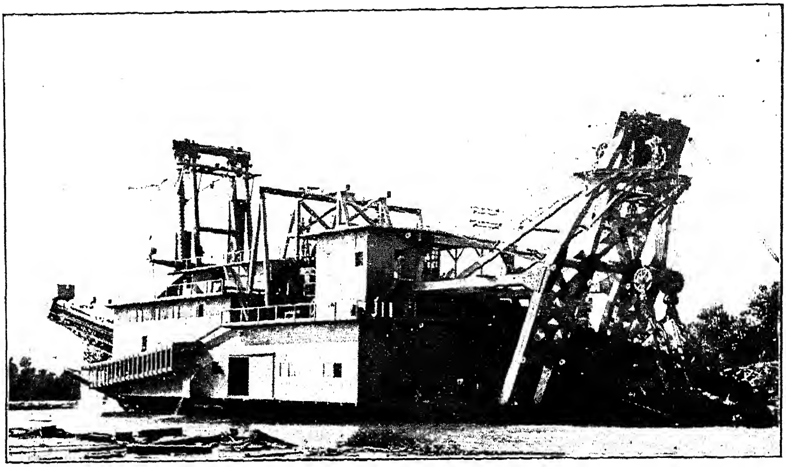


FIG. 2.—YUBA NO. 13, A 15-CU. FT. DREDGE.

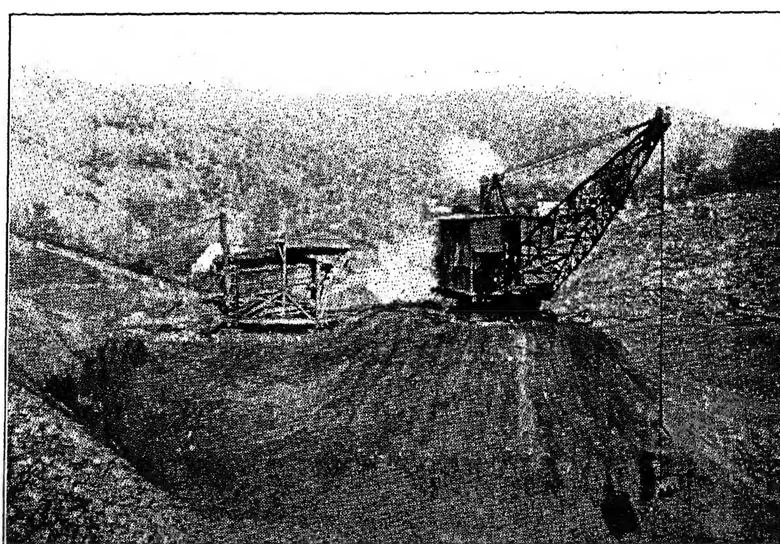


FIG. 3.—BUCKET-SCRAPER PLANT AT WORK.

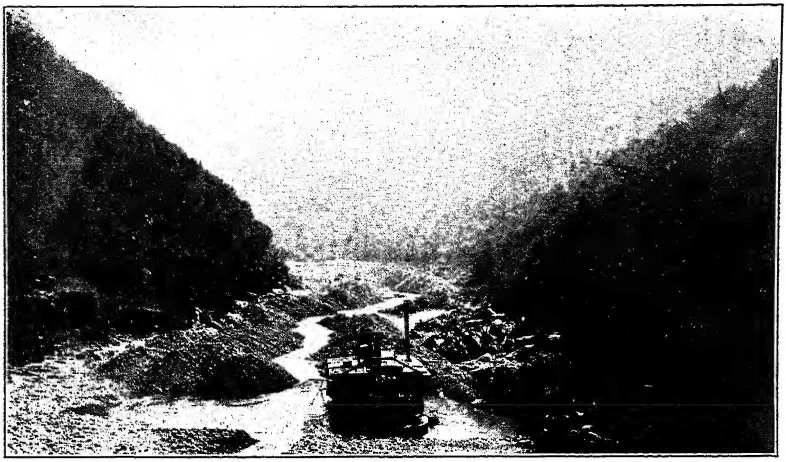
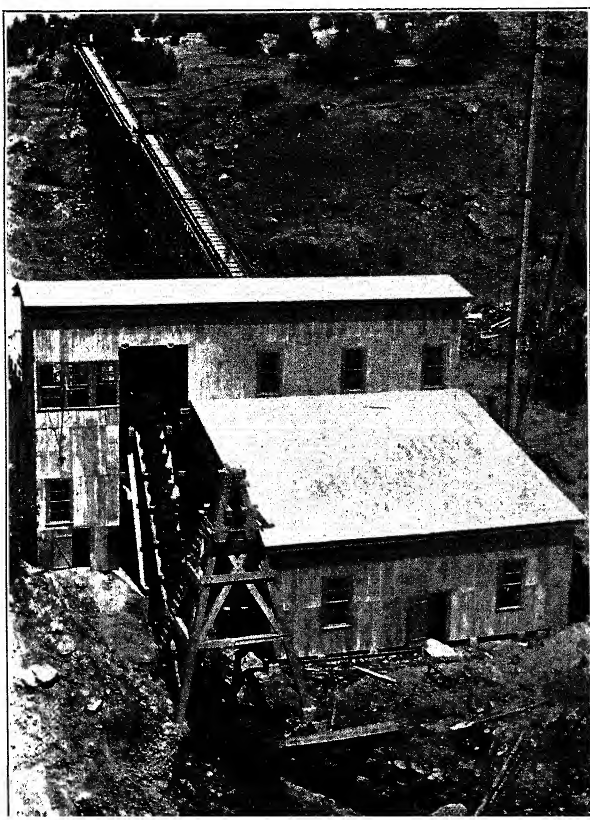


Fig. 4.—3.5-ft. Risdon Dredge Operating in the American River.



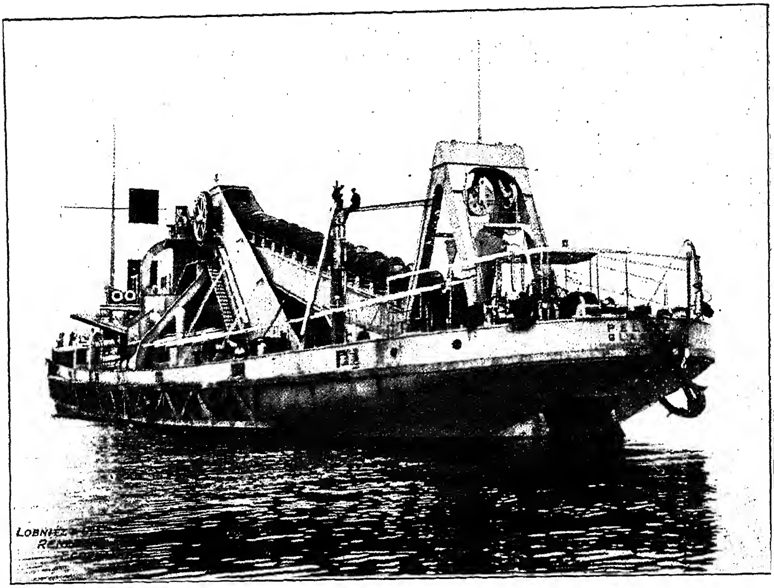


FIG. 6.—HARBOR-DREDGE; LARGEST BUCKET-DREDGE AFLOAT; 54-CU. FT. BUCKETS.

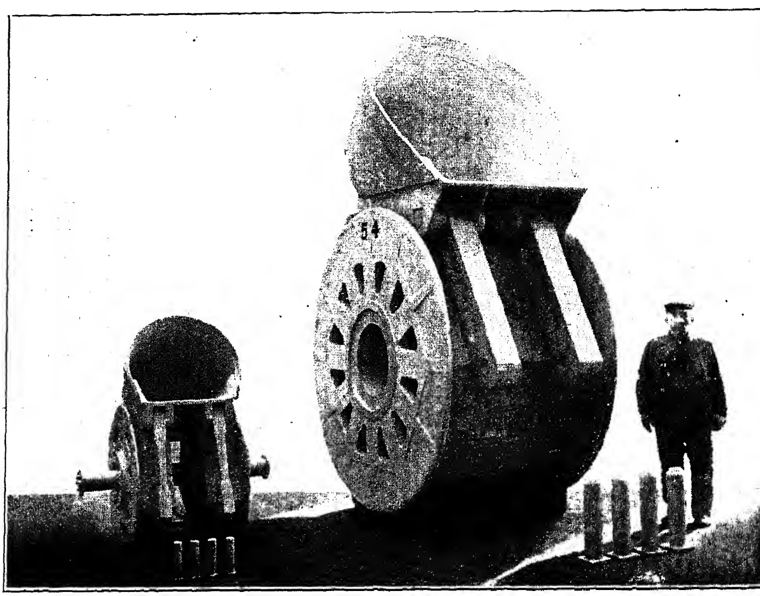


FIG. 7.—5-CU. FT. BUCKET AND TUMBLER COMPARED TO A 54-CU. FT. BUCKE FROM A HARBOR-DREDGE. (Lobnitz & Co., Renfrew, Scotland.)

fornia, the average depth of gravel, and the value per cubic yard. Much of this ground has already been dredged, and some areas of lower-grade gravels which ultimately may be dredged are not included.

TABLE II.—*Dredging-Ground in California.*

Counties.	Total Proved Dredging-Ground	Average Depth of Ground.	Average Value Per Cubic Yard
	Acres.	Feet.	Cents
Butte	6,600	30	15
Yuba	3,600	65	15
Placer	430	38	8
Sacramento	6,000	35	11
Calaveras	850	18	14
Stanislaus	200	22	14
Merced	400	20	13
Shasta	600	22	11
Siskiyou	350	35	14
Trinity	600	25	15

In addition to the lower-grade gravel being worked in the future, areas considered too small for the profitable installation of an expensive modern dredge will be equipped with strong, lighter designed, and less expensive boats, and also with rebuilt dredges using machinery from dredges which have worked out the areas for which they were built, or that have been dismantled and replaced by larger boats. The machinery of some of these dismantled, and to be dismantled, dredges is in good condition, fit for many years of working-life, and can be refitted on a new hull on nearby property or properties not too difficult of access, and a practically new dredge built, in some cases at less than 50 per cent. of the cost of the original boat. It must be recognized that these rebuilt boats may not always be adapted to handle the gravel with as low operating-costs as might otherwise be attained, but the smaller expense of installation will prove a large factor in their selection and use. Dredges that were first constructed in Colorado, but proved unprofitable, were dismantled and their machinery used on hulls in California. The machinery from several California dredges has been moved to other fields, and, in some cases, to Alaska. Recent examples which may be mentioned are the Scott River dredge, formerly at Callahan, Siskiyou county, where it was unprofitable, which was dismantled and the ma-

chinery moved to Trinity Center, Trinity county. The Butte dredge, having worked out the company's holdings at Oroville, was also dismantled, and the machinery is being placed on a new hull near Jenny Lind, Calaveras county. The Scott River dredge was put in commission in August, 1908, and was equipped with 7.5-cu. ft. buckets. It was not quite two years in operation, being shut down May 30, 1910. It was purchased by the Alta Bert Gold Dredging Co., acting on the advice of H. G. Peake, and was moved to its ground in Trinity county. The estimated cost of building a new hull, installing the machinery, including a 28-mile haul, with a freight-cost exceeding 1 cent per pound, and building a power-transmission line of 5 miles, is $80,000. The Butte dredge was put in operation during November, 1902, and dismantled in July, 1910. It was equipped with 3.5-cu. ft. buckets. The machinery is being placed on a new hull, and includes a new bucket-line of 4-cu. ft. buckets. The cost of the installation, including the new bucket-line, has been estimated at $30,000. The figures given for moving both of these boats must be considered approximations only, as they are not official.

There also seems to be a field in California and elsewhere for the installation of the bucket-scraper on auriferous areas too small or otherwise unsuitable for dredges, but of sufficient gold-content to be profitably handled by the scraper. This method of handling gravel is profitably in use in Siberia, in the Kolchan mines, at the present time, a plant built by the New York Engineering Co. having been installed by C. W. Purington. A view of a bucket-scraper plant is given in Fig. 3. In California one has been in successful operation near San Andreas. This machine rests upon rollers, by which it is moved on a plank track. It delivers to a set of trommels and gold-saving tables similar to those on a dredge. It has a 60-ft. boom upon which the scraper-bucket, weighing 1.5 tons and having a capacity of 1.5 cu. yd., works. The bucket is raised and lowered by means of a cable working over a sheave at the end of the boom, and is loaded by means of a drag-line traveling between sheaves in front of the floor-plate. Dumping is accomplished by means of an equalizing-cable attached to the drag-line and on the front of the bucket, which passes over a sheave fastened to the bucket-bale. The exca-

vator is turned by a single-drum winding-engine, having two cables attached, whereby a complete circle can be made and the scraper-bucket operated on all sides. The machine is operated by steam-power, wood being used as fuel. Thus equipped, it has excavated gravel to a depth of 35 ft., and, it is claimed, can be worked to a depth of 50 feet.

The material is dumped by the bucket into a hopper 12 by 12 ft., which feeds a trommel-screen 4.5 by 22 ft., the upper part of which has $\frac{3}{8}$-in. perforations, the perforations of the lower 18 in. being 0.75 in. The oversize discharges to a belt-conveyor stacker; the undersize passes over Hungarian riffles, and then to a riffled sluice-box in which quicksilver is deposited, and finally to a 20-ft. sluice-way in which cocoa matting is used. Water is pumped into the hopper to wash the material through the cylinder. The cylinder and stacker are operated by a 15-h-p. electric motor, and the whole washing-apparatus is mounted on rails. It requires two men on the excavator and one on the washer. Accurate figures of operating-costs are not at present available, but are understood to approximate 16 cents per cubic yard. At the Kolchan mines it is claimed that exclusive of management-charges, which are high, the cost of washing 24,400 cu. yd. for July was 14 cents per cubic yard. While these machines cannot be compared with modern dredges in capacity and operating-costs, it is claimed by those familiar with the operation that there is a good field under suitable conditions for their use in places where it is impracticable to install dredges.

The dipper-dredge has been successfully operated on small areas at Oroville and elsewhere, but does not meet with approval among dredge-operators in general, who contend that the efficiency of these boats, both as to yardage and gold-saving capacity, is not up to that of the standard type. These boats have a low first-cost (about $25,000, f.o.b. factory) and are built with buckets of from 1.25- to 2.5-cu. yd. capacity. It is claimed by the dealers and some operators that under the following conditions there is a field for this type of dredge: (1) where the ground is somewhat shallow; (2) where the extent of the ground is not sufficient to warrant the installation of a costly dredge; (3) where the material is of a rough character, boulders, and stumps; (4) where the ground is mixed with

more or less clay, as the dipper will relieve itself notwithstanding the adhesiveness of the material.

The reported successful operation of a small Risdon dredge on the middle fork of the American river near Forest Hill, Placer county, under conditions thought by many to be impossible for operation, will undoubtedly encourage other installations in rivers at times torrential. A. A. Tregidgo is now promoting a company for the dredging of gravel some distance below this place. Without attempting to pass on the merits of either of these undertakings, it is interesting to consider them as engineering problems, and their success will draw considerable attention to similar gravel-areas in this State and elsewhere. While gold-dredging in California has been mainly confined to gravel-areas some distance from the main river-channels, it is claimed that a small boat, with some modifications in the hull to suit the river conditions, and adapted for work in a swift current, with head-line and mooring-winches of greatly increased strength, can be profitably operated, even in the winter months, in the California rivers where not in conflict with the present dêbris laws. In addition to the use of a small dredge, it is proposed by Mr. Tregidgo to operate a hydraulic elevator on the same property, water being available at a head of 1,000 ft. This water will first be used at a head of 400 ft. to generate electric power to be transmitted to the dredge. From this point the water will have a head of 600 ft., to be used in the hydraulic elevator. In addition to these enterprises, there are several proposed dredge-installations on somewhat similar areas in this State, concerning which definite information is not available at present. Fig. 4 is a view of a 3.5-ft. Risdon dredge operating in the American river.

The suction-dredge has never been favorably considered in gold-dredging, except by the inventors and builders. It is claimed by those interested that one is in successful operation in Shasta county, and another in Siskiyou county, though other information is to the effect that these boats were not a financial success and are no longer operating.

A method closely allied to dredging, which may be termed a hybrid of dredge and hydraulic mining, is attracting much attention in California. This is the plant of the Tarr Mining Co. at Smartsville, which was built to operate the old Blue Gravel hy-

draulic mine. Fig. 5 is a view of the plant. This mine was a producer in early days, but was shut down by the Débris Commission. This company believes that it will be able to operate in compliance with the present law. From an engineering standpoint, the proposal has some interesting features. Briefly, it consists of hydraulicking the gravel-bank to a sump in front of a stationary dredge-building of concrete and sheet-iron, where a regular steel-girder dredge-ladder, equipped with fifty-two 7-cu. ft. buckets elevates the gravel to a trommel 45 by 6 ft. with 0.5-in. holes. From the screen the undersize flows to gold-saving tables with Hungarian riffles having an approximate area of 4,600 sq. ft. The oversize passes to a belt-conveyor 570 ft. long, built in two sections, each section being driven by a 50-h-p. motor. A 100-h-p. motor is used on the digging-ladder, and a 30-h-p. motor on the revolving-screen. At the end of the belt-conveyor stacker two Bleichert tramways are being constructed. These will afford a much larger dumping-ground for the tailings.

The fine material, after passing over the gold-saving tables, flows through a bed-rock tunnel about 0.5 mile long and is elevated to a concentrating-plant equipped with tables of the Overstrom type. The material first passes through revolving-screens, the oversize being carried outside the concentrator, and the undersize to the tables. It is the idea of the management that this plant will save black sand, which is claimed to be valuable, and any gold and platinum that escapes the first tables.

The concentrator stands several hundred feet from the Yuba river, and a concrete dam will be constructed to afford a settling-basin for the tailings. This experiment will be watched with interest. Its success will undoubtedly mean that other properties formerly worked as hydraulic mines, which have been shut down by the Débris Commission, will be operated on somewhat similar lines. The equipment of such a property is no small matter. The operating-cost as yet is purely speculative. The management of the Tarr company does not believe that the cost of operating the plant will exceed 8 cents per cubic yard.

On Bonanza creek, in the Yukon, a portable bucket-elevator arranged to elevate gold-bearing gravel to a system of portable sluices, the position of which can be changed when necessary

to obtain a new dump, has been in more or less successful operation by the Yukon Gold Co. for a number of years, but only one attempt, so far as is known, has been made to adjust this method to California gravels. The mode of operation is as follows: A sump approximately 20 ft. square, with a depth of from 14 to 16 ft. below bed-rock, is excavated to receive the lower end of the elevator. A channel or bed-rock sluice emptying into the sump, with a grade of 5 in. in 12 ft., is excavated in the bed-rock and provided with riffles. The gravel-bank to be treated is hydraulicked with two 3-in. giants, and a third giant sluices the gravel to the sump, from which the buckets elevate it to a riffled sluice about 25 ft. high. The elevator-ladder is equipped with buckets of 3 cu. ft. capacity, close-connected, and driven by a 50-h-p. motor. The water used in the upper sluice is pumped from the sump by one 12-in. centrifugal pump belted to a 100-h-p. motor, and one 8-in. pump driven by a 50-h-p. motor. A derrick with a long boom is placed in a position convenient for handling any large boulders. Records of operating-cost have not been made public by the Yukon Gold Co., and it is understood that the use of these machines will be discontinued or considerable changes made in the method of operating them.

A somewhat similar machine was operated a few months during 1910 at Poker Bar, Trinity county. This was installed by R. E. Whitcomb, at a cost of approximately $15,000. The motive-power was steam, wood being used as fuel. The expenses of operation were great, but no accurate data are obtainable at present. It is said that the operation of the machinery thoroughly demonstrated the value of the gravel-area, and it is reported that a dredge will be installed this year. The management contemplates moving a Marion dipper-dredge, formerly successfully operated at Oroville, and which had turned over the holdings of the original company. It is estimated that this dredge can be put in operation at a cost of $15,000. At the present time there are 62 bucket-elevator dredges operating in California, and five under construction. Of the six dredges put in commission in 1911, four have been built by the Yuba Construction Co. and are equipped with Bucyrus machinery and 15-cu. ft. buckets, one was built by the Union Iron Works and equipped with 8.5-cu. ft. buckets, and one by the Risdon

Iron Works with 4-cu. ft. buckets. One of those under construction has buckets of 15 cu. ft. capacity, one 7.5-, one 7-, one 5-, and one 4-cu. ft. buckets.

It is interesting to note that of the 62 dredges, which are operated by 28 companies, 30 are operated by three companies controlled by W. P. Hammon and associates, distributed among three counties, as follows: Butte county, 8; Yuba county, 13; Sacramento county, 9. It may here be mentioned that the great progress and improvement is due in a great measure to the enterprise and successful operations of Mr. Hammon and his associates. Couch dredge No. 1, the first successful bucket-elevator dredge put in commission in the State, was financed by Mr. Hammon and the late Thomas Couch, and it seems eminently fitting that Mr. Hammon should be the leading gold-dredging operator in California, and in control of the largest dredging companies in America.

What seems to be a record in dredge-construction and worthy of mention is the building of the dredge for the Julian Gold Mining & Dredging Co. on Osbourn creek, near Nome, Alaska. This dredge was constructed by the Union Construction Co., of San Francisco. The dredge was shipped from San Francisco on June 1, arriving at Nome June 13. On June 17 the company commenced hauling material, and on July 22 the dredge was completed and operations started. The dredge-hull is 30 by 60 by 6.5 ft. It is equipped with 34 open-connected 2.75-cu. ft. buckets, and is designed to dig 14 ft. below water-level. Power is furnished by gasoline-engines as follows: one 50-h-p. for digging-ladder, winches, and screen; one 30-h-p. for pump; one 7-h-p. for lighting apparatus; a total of 87 h-p. Distillate costs at Nome 21 cents per gallon. Operating-expenses at present range from $110 to $125 per day, and the capacity of the dredge is from 1,000 to 1,300 cu. yd. per day, indicating an operating-cost of from 10 to 11 cents per cubic yard, exclusive of repairs. The cost of the dredge complete and in operation was $45,000. The Union Construction Co. also built a similar dredge for dredging tin, near Cape York, this latter being the first tin-dredging operation to be carried on in America. Its future will be watched with interest and may be followed by further installations.

With the development of the gold-dredge to its present effi-

ciency, the question is often raised as to when the limit in size for economic dredge-installation will be reached. Much depends upon the conditions met in operation. There is no question as to the mechanical possibility of larger buckets. In Boston harbor a bucket-elevator dredge equipped with buckets of 2 cu. yd. capacity has been successfully operating for some years on harbor-work, and on the Danube river in Germany a bucket-elevator dredge having 2.5-cu. yd. buckets is now in operation. Fig. 6 is a view of a harbor-dredge equipped with 2-yd. buckets, and a 5-ft. and a 54-ft. bucket are shown in Fig. 7. While the mechanical possibilities have thus been proved, to apply such radical changes in size to the gold-dredge of to-day would necessitate an entirely different arrangement of the gold-saving tables and would probably result in a general modification of the whole gravel-washing apparatus now in use. Even the most optimistic advocates for increasing the size of the dredge-buckets would hesitate at recommending a 2-cu. yd. bucket, which is nearly four times the present size of the buckets on the largest gold-dredges in operation, but there are a number of engineers who believe that the bucket-elevator dredge with buckets having a capacity of 1 cu. yd. will be constructed before long. While a dredge of this character would necessarily be equipped with heavier machinery and a larger hull than those on the present 15-cu. ft. boats, it is, as before stated, quite possible that, with modifications of the washing-apparatus, the hull of the 1-cu. yd. dredge may not be proportionately larger. The present 15-cu. ft. boats have a hull 60 by 150 by 12.5 ft., with a deck overhang of 5 ft. on either side, making a total width of 70 ft. The gold-saving tables are of the double-bank type and have an approximate area of 7,000 to 8,000 sq. ft. Without some change in the washing-apparatus, it can readily be seen that 14,000 sq. ft. of table-area would either necessitate a hull of greatly increased size, or additional tiers of tables, for which an increased length of bucket-ladder would be required to elevate the gravel to the additional height, or a general change in the design of the boat. Practice has demonstrated that when digging free-washing gravel the table-area of the 15-ft. boats is considerably in excess of all requirements, and some operators contend that it would not be neces-

sary to increase the table-area proportionally when buckets of 1 cu. yd. capacity are constructed.

There may be a field for dredges of this size, for instance, in the Oroville and Folsom fields, to re-dredge the tailings-piles left from the first dredging-operations. After many of the cobbles have been removed for the rock-crushing plants, the ground, if dredged, will, in many cases, yield a fair return. Especially would this be the case in the areas where the early dredges worked, as the gold-saving apparatus of the first successful dredges was not as efficient as that in present use.

In addition to the gold recovered from the gravel, the reclamation of the land for agricultural purposes might be a considerable factor in estimating the total profit to be won from the installation of a mammoth dredge for this class of work. The first dredges, in turning over the ground, necessarily deposit the top soil on the bottom, and the gravel and boulders from the tailings-stacker on top of this. After much of the coarser gravel is removed for rock-crushing operations, with some such arrangement as that which is being tried out in New Zealand in re-soiling experiments, this soil now below the gravel could, in re-dredging, to a great extent be deposited on the top of the coarser material. In reclaiming the dredged land it is, no doubt, a matter worthy of consideration. In this connection it is interesting to note the successful experiments of the Natomas Consolidated and others in planting eucalyptus trees on dredged land after the larger gravel has been removed, no re-soiling being necessary. Any estimate of the operating-cost of a dredge of this character is, of course, pure speculation, but there seems every reason to expect that, under favorable conditions, or in re-dredging some of these previously-dredged areas, a very low operating-cost would be obtained.

The operating-cost of dredging is always a matter of interest, but working-costs cannot be fairly used in comparison unless uniform methods of determining them are employed, and also unless operating-conditions are somewhat similar. As in other branches of the mining industry, it may also be said that the apparent operating-cost is in a great measure a matter of bookkeeping. As the time available for preparing this article was limited, it has been found impossible to prepare new data on working-costs of dredges in California, so I have utilized a table

prepared last year by me (Table III.).[3] Under similar conditions, the operating-costs are practically the same. The new boats have not been working long enough to make any figures of operating-cost of much value, but it is understood that they will under the same conditions appreciably lower the costs obtained by the 13.5-ft. boat.

It is interesting to note the following average operating-cost per cubic yard of the large companies working in California during 1910. The Yuba Construction Co., for the year ended Feb. 28, 1911, handled 13,970,728 cu. yd. at a total cost of 5.67 cents per cubic yard. The Natomas Consolidated handled, for the year ended Dec. 31, 1910, a total of 15,989,525 cu. yd. at a total cost of 4.52 cents per cubic yard, and during the six months ended June 30, 1911, a total of 10,793,891 cu. yd. at a total operating-cost of 3.78 cents per cubic yard. This company has put in commission during this year three dredges with buckets having a capacity of 15 cu. ft., one being in the Feather River division at Thermalito, and two in the Folsom division on Rebel hill. These two boats are now satisfactorily handling ground that for a long time was considered too difficult for economical dredging. The gravel is deeper and more compact than any other in the district, and dredge No. 8 is handling ground containing much stiff clay. The Oroville Dredging, Ltd., for the year ended July 31, 1910, handled 5,661,612 cu. yd. at a total cost of 5.05 cents per cubic yard.

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[3] *Mining and Scientific Press*, vol. ci., No. 5, p. 151 (July 30, 1910).

TABLE III.—*Working-Costs of Gold-Dredging in California.*

Capacity of Buckets.	Time in Commission.	Working Period for Figures Given.	Actual Working-Time Hours During Working Period.[a]	Quantity Handled.	Average Depth of Gravel	Operating-Expenses, in Cts. Per Cubic Yard. Labor and Material	Electric Power.	Water.	Repairs.	General.	Taxes and Insurance.	Total Expense.	Remarks
Cu. Ft			Hrs.	Cu. Yd.	Ft.	Cts.	Cts.	Cts.	Cts.	Cts	Cts	Cts.	
3	....	1 yr.	2,809	173,655	27.0	2.77	0.90	0.14	4.15	0 78	0.49	9 23	Difficult digging [g]
3	5 yr. 9 mo.	1 yr.	7,216	458,882	26.9	2.03	0.69	. ....	3 28	0 63	0 37	7 00	Working under favorable conditions.
3.5	7 yr	1 yr.	..	395,316	35.0	2.83	1 53	0.228	1.74	1 32	....	7.67	
3.5	6 yr. 6 mo.	1 yr.	7,344	461,882	35.0	2.85	1.48	0.195	1.71	1 07	... .	7.32	Compact gravel land subject to overflow.
4	9 yr.	1 yr.	7,057	484,387	20.6	1.83	0.89	0.31	2.58	0 65	0.26	6.52	Remodeled dredge, uneven bed-rock, in places shallow
5	6 yr.	1 yr.	. ....	481,184	25.0	3.28	1.46	0.39	2.97	1.52	....	9.55	Difficult ground, in places cemented gravel.
5	2 yr. 5 mo.	1 yr.	. ...	635,146	27 0	3 14	1.45	. . ..	2.40	1.28	0 41	8 70	Difficult ground.
5	2 yr.	1 yr.	..	582,891	30.0	3.28	2.02	0.32	2.59	1.37	1.00	9.60	Difficult digging.
5	5 yr. 6 mo.	1 yr.	7,344	615,009	25.0	3.06	1.42	0.29	3.06	1.14	..	8 98	Difficult digging
5	4 yr. 6 mo.	1 yr.	..... .	812,355	36.0	2.30[b]	1 08	.. . ...	2.95	.	0 35	6.65	Medium gravel with considerable clay, much brush on top soil
5	3 yr 5 mo.	1 yr.	6,798	1,148,480	25.5	0.88	0.52	0.05	1 77	0 25	0 35	3 80	Loose gravel, heavy overburden of sandy loam.
5	2 yr. 5 mo.	1 yr.	6,790	1,148,802	29.9	0.82	0.49	0.03	1.89	0.25	0 16	3 64	Loose gravel, heavy overburden of sandy loam
5	4 yr. 7 mo.	1 yr.	6,644	599,614	38.5	1.77	0.92	0 25	4.03[c]	0.47	0 28	7.67	Difficult digging, working against 20-ft bank
7	9 mo. 10 days.	1 yr.	5,088	838,885	35.0	1.19	0 69	... . .	1 22	0 26	0.17	3 53	Difficult digging, gravel coarse, partly cemented [h]
7	1 yr.	1 yr.	6,313	1,114,605	27.6	1.21	0 62	0.03	1 81	0 29	0 11	4 07	Compact gravel
7	3 yr. 9 mo.	1 yr.	6,390	1,033,694	26.5	1.08	0.64	0.14	2.69[d]	0 34	0 20	5.09	Compact gravel, heavy digging
7	2 yr. 9 mo.	1 yr.	6,917	1,017,167	28.1	1.10	0.65	0.15	2.19[e]	0 28	0 14	4 51	Compact gravel, heavy digging
7	3 yr.	1 yr.	6,352	935,322	33.4	1.26	0.85	0 06	3 06	0.31	0 34	5 88	Compact gravel.
7	3 yr.	1 yr.	6,700	1,194,146	27 5	1 05	0.58	... .	2 78	0.32	0 37	5.10	Compact gravel.
7.5	2 yr. 11 mo.	2 yr. 11 mo.	13,464	3,458,229	27 9	0 27	0.41	... .. .	1 50	0 24	0 24	4.42[f]	Medium compact bench gravel.
7.5	9 mo 6 days	9 mo. 6 days	5,582	944,879	28.9	0.95	0.58	.... ..	1.30	0.27	0 39	3 55	Medium compact gravel with heavy over-burden.
7.5	2 yr. 6 mo.	1 yr.	6,402	1,369,844	70.2	0.99	0.77	....	1.95	0.45	... .	4.16	Medium gravel overlain with hydraulic tailings.
7.5	2 yr. 6 mo	1 yr.	6,900	1,281,351	67.8	1.09	0.98	....	2.01	0.45	....	4 53	Medium gravel overlain with hydraulic tailings.
8	6 mo.	6 mo.	3,162	583,927	42.5	1.69	0.59	.. ..	1 14	0.28	0 22	3 92	Light gravel, dredge working against 10-ft bank.
8	4 mo. 8 days.	4 mo. 8 days.	2,369	626,624	24.0	Segre	gated	costs	not	given		2 47	
9	5 mo.	5 mo.	....	580,310	51.0	Segre	gated	costs	not	given		4 98	Cemented gravel, difficult digging, 20-ft bank above water-level.
13 5	8 mo.	8 mo.	4,478	1,803,201	19.0	1.02	0.47	... ...	0 60	0.12	0.09	2 30	Fine gravel, easy digging.

a. Total possible time in year's work, 8,784 hours.
b. Including general expense, management, etc.
c. Heavy repair-cost due to new tumbler, conveyor-belt, repairs to digging-ladder, screens, etc.
d. Replacing tumbler-shafts, conveyor-belt, and new screen included in repairs.
e. New steel spud and screen in repairs.
f. Depreciation-charges included in total expense.
g A 7-ft. dredge is now working this ground at a profit.
h. This dredge successfully replaced an open-connected bucket-dredge which could not handle ground at a profit.

Electrolytic Refining at the U. S. Mint, San Francisco, Cal.

BY EDWARD B. DURHAM, E. M., BERKELEY, CAL.

(San Francisco Meeting, October, 1911.)

THE refinery at the San Francisco Mint takes the bullion purchased by the receiving department, and carrying more than 200 parts of precious metals in 1,000, or, in mint parlance, over 200 fine, and separates and refines the various metals contained therein, using electrolytic processes exclusively.

Bullion containing silver is treated in cells charged with a nitric electrolyte. These cells produce fine silver and leave a residue rich in gold.

The residue from the silver-cells, together with crude gold-bullion, is treated in cells having a chloride electrolyte. These produce fine gold and leave a residue containing silver chloride. The latter is reduced to the metallic state with zinc and is then treated in the silver-cells.

The various waste solutions and the wash-waters, after being freed from the bulk of their precious metals, still contain copper and other metals. These are removed by scrap-iron, and are then treated in the copper-cells, having a sulphate electrolyte. These cells produce pure copper, and collect a residue containing lead, some gold and silver, and all the metals of the platinum group that were in the bullion. This residue is relatively small, and is melted into bars and stored until sufficient accumulates to warrant treating it for platinum, etc.

The refinery occupies three large and three small rooms. The large ones are, a melting-room, 30 by 34 ft.; a cell-room, 39 by 46 ft.; and a wash-room, 30 by 33 ft. The small rooms are used as foreman's office, laboratory, and generator-room, respectively.

The methods here described are those in use in December, 1909, when notes for the present paper were taken.

I. Silver-Refining.

An outline of the system is shown by the diagram, Fig. 1, which gives the order of events and the interdependence of the various operations in a brief form.

1. *The Apparatus.*

A. The Anodes are made up of crude silver-bullion, together with gold-bullion that is too low in gold to be easily made up

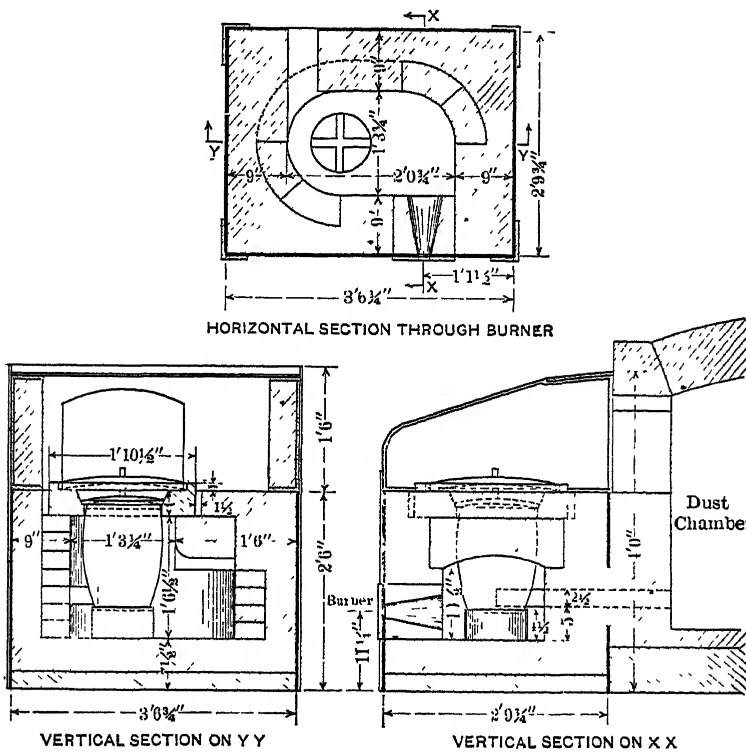

Fig 2.—Melting-Furnace.

into gold anodes. The endeavor is to make a mixture, such that the anodes will run about 600 thousandths in silver, 300 thousandths in gold, and the remaining 100 thousandths in base metals. The metal is melted in No. 100 graphite crucibles, in Rockwell melting-furnaces of the "open-top mint type," heated with crude oil. A drawing of these furnaces is given in Fig. 2. The furnaces are used for melting both the crude metals for

the anodes, and the fine gold- and silver-products of the refinery that are to be cast into bars. Fig. 3 is a view of the melting-room. In the background are the furnaces; in the foreground, to the left, is a truck-load of anodes; in the center a truck loaded with gold bars (dark), and behind it a truck loaded with silver bars (white).

The anodes are cast in open cast-iron molds, and are of the dimensions given in Fig. 4. They are suspended from the conductors by C-shaped hooks of gold, which pass through the hole at the top of the anodes and over bars which form the conductors for the current. The anodes are immersed for their full depth in the electrolyte.

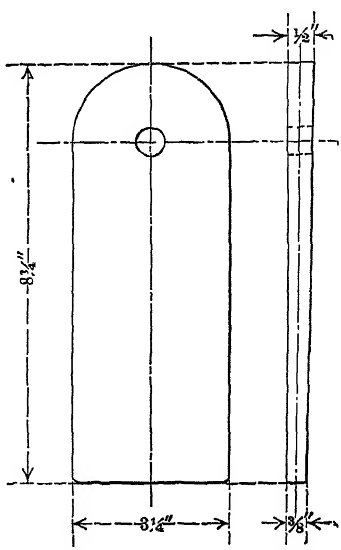

FIG. 4.—DIMENSIONS OF ANODE FOR BOTH THE SILVER- AND THE GOLD-CELLS.

B. The Cathodes are made of sheets of silver, 1000 fine, 0.051 in. thick (No. 16 B. & S. gauge) and 4 in. wide. They are immersed for 8.5 in. in the electrolyte, and are bent over at the top so as to hang from the conductors.

The crystallized silver that collects on the cathodes is loose and is removed daily. To facilitate this stripping, the cathode sheets are treated with a "dope," consisting of silver nitrate, copper nitrate, and hydrochloric acid, all mixed together, and painted on with a rag. The sheets are then dried in the dry-

room. One dose of this dope lasts two or three months; then the deposits begin to stick, and the plates are re-treated.

C. The Electrolyte consists of water with 3 per cent. of silver, as silver nitrate, from 1.5 to 2.5 of free nitric acid, and a little glue. The latter is dissolved and poured in as a thick liquid. The effect of the glue is to toughen the deposit of silver on the cathode.

The electrolyte dissolves and retains the copper and other soluble base metals. These do no harm until the solution becomes so strong that the purity of the silver deposited on the cathodes is affected, when it has to be changed.

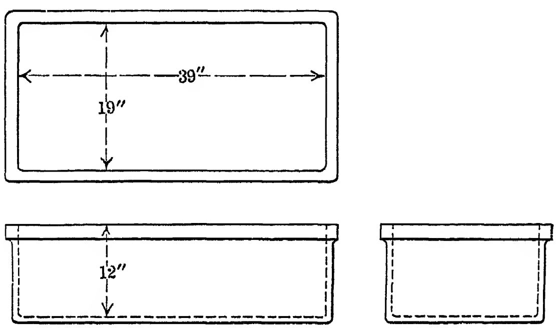

FIG. 5.—SILVER-CELL, OF BROWN EARTHENWARE, USED FOR BOTH THE HORIZONTAL AND THE VERTICAL PROCESSES.

D. The Cells are of brown earthenware and their dimensions are shown in Fig. 5. Experience has shown that they are too shallow for advantageous work. There is only a small space between the bottom of the cell and the lower end of the anodes, and the slimes that collect in this space soon cause short-circuits which stop the action of the cell. A new set of cells, 18 in. deep inside, instead of 12 in., is about to be installed. These deeper cells will allow longer cathodes to be used, and, since the cores that have to be re-treated will be of the same size, there will be a reduction in the percentage of metal to be re-treated.

The cells are placed end to end in a double row on two long benches, 12 on one bench and 6 on the other. This allows all the cells to be easily inspected and attended to, from one side or the other of the benches. These cells are the dark ones on the second and third benches in Fig. 6.

The anodes and cathodes are hung in alternate rows from maple strips, $2\frac{1}{8}$ in. apart from center to center, which extend across the cells. Along the top of each is laid a gold strip, bent into the form of an inverted trough. These gold strips are connected by screws alternately to the positive and negative bus-bars, and form the conductors. There are 19 of these across each cell, 10 supporting four cathodes each and 9 supporting four anodes each. The bus-bars are of copper and extend along the main wooden frame that covers the top of the entire bench of cells. All woodwork and the copper bars are coated with "biturine solution," an asphaltic paint that comes from Australia, to protect them from the action of the acids.

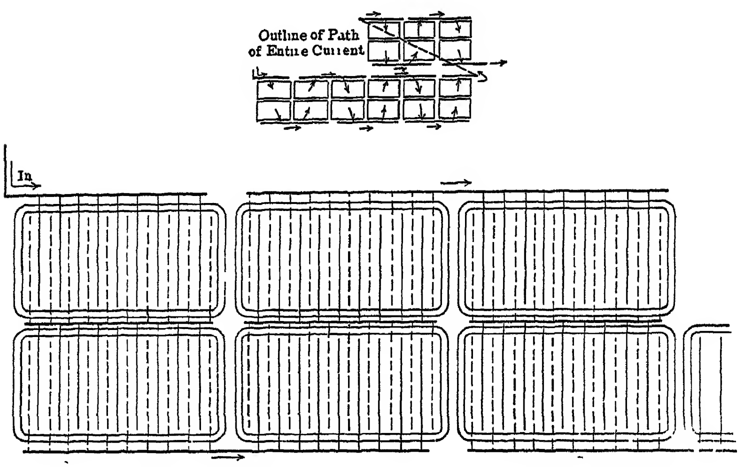

Fig. 7.—Diagram of Path of Current through the Vertical Silver-Cells.

The solution in the cells is kept in motion by two glass propellers in each cell. This prevents the heavier solutions from settling to the bottom, and makes the deposition uniform over the whole cathode.

Each propeller, 2 in. across, is made in one piece with a glass rod, which runs up vertically between the electrodes, and is driven by a cord running in a grooved pulley at its top. The vertical glass rods, as well as the line-shaft, are carried by a wooden frame above the cells, as shown in Fig. 6.

E. The Current is a direct one of 15 volts, and passes through the 18 cells in series, as shown in Fig. 7. The amount of

current is such as to give a density of 8.3 amperes per square foot of cathode-surface. There are 40 cathodes per cell and each has a normal immersion of 8.5 in. The end rows of cathodes have only one effective surface, so the total cathode-surface per cell is:

$$(2 \times 8 \times 4) + (2 \times 4) = 72 \text{ surfaces, or } \frac{72 \times 4 \times 8.5}{144} = 17 \text{ sq. ft.}$$

The total current required is therefore $17 \times 8.3 = 141$ amperes.

Fig. 8 is a view of the generator-room and shows the machines and the switch-board. The generators are driven by current obtained from a public power-line and furnish direct current of the required potential for the different operations.

F. Centrifugal Machines are used to separate the moisture from the different products of the refining process, and to wash them free from soluble matter. There are two of these machines. No. 1 belongs primarily to the silver process, and is used exclusively for silver or products charged with nitric compounds. No material containing chlorides is ever placed in it. Centrifugal No. 2 is similar to No 1, but is reserved for the gold process and for solutions carrying chlorides.

The rotors of the centrifugals are of earthenware and provided with ducts for the escape of the liquids. When in use, the rotor is lined with one thickness of 7-oz. duck, and in this bag is placed the material to be treated. A different filter-bag is kept for each different kind of material that is washed.

All the products of the silver process can be dried sufficiently in the centrifugals, so that they can be transferred to the crucibles and melted.

Fig. 9, a view of the wash-room, shows the centrifugals with their driving motors.

2. *Operation and Products.*

Briefly, the anodes are dissolved; pure silver collects on the cathodes; copper and other metals forming soluble nitrates go into the bath, and gold and other insoluble metals are left as a sponge on the anodes.

As the dissolving action progresses, the anodes are taken out at intervals and the sponge of insoluble metals is shaken off

into an earthenware jar, by knocking them against its sides. This spongy material is crude or black gold with about 10 per cent. of silver and 1 per cent. of base metals. After washing in centrifugal machine No. 2, it is melted into anodes for the gold process.

When the anodes are eaten down so that they barely hold together (which takes about 48 hr), they are removed, all the loose spongy material is knocked off, and the hard cores that remain are treated in the horizontal cells, to be described later. New anodes are then hung in their places.

So long as the electrolyte contains an ample supply of silver, this is deposited in preference to the base metals.

The electrolyte is tested at intervals to determine its strength in silver, and if this test shows that the bath is too low in silver, its strength is brought up by adding strong silver nitrate solution.

The test for silver is made by gradually adding a standardized solution of ammonium thiocyanate, NH_4SCN, to a sample of the bath, a little ferric sulphate solution having been previously added as an indicator. When all the silver has been precipitated, the ferric salt gives a red color. This is Volhard's method, and is given in detail by Sutton.[1]

When the bath contains about 8 per cent. of copper it has to be changed, since the silver deposited on the cathodes begins to be contaminated with the copper. This spent electrolyte is treated in the scrap-copper tank to recover the silver, and then passes on to the scrap-iron tank, where the other metals contained in it are caught, as will be described under the head of Copper-Refining.

The pure silver collects in a crystalline condition on the cathodes, which are lifted out daily and cleaned over large porcelain jars. At first, the deposit is loose and fern-like, and most of it can be removed by knocking the cathodes against the sides of the jars. Gradually a firmer deposit collects that will not knock off, and this has to be removed with a scraper, when it comes away in sheets and leaves the cathode entirely clean. This pure silver is washed in centrifugal machine No. 1 until free from acid and soluble salts, and then is whirled until dry enough for melting, when it is made into fine bars.

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[1] *Volumetric Analysis*, 7th ed., p. 142 (1896).

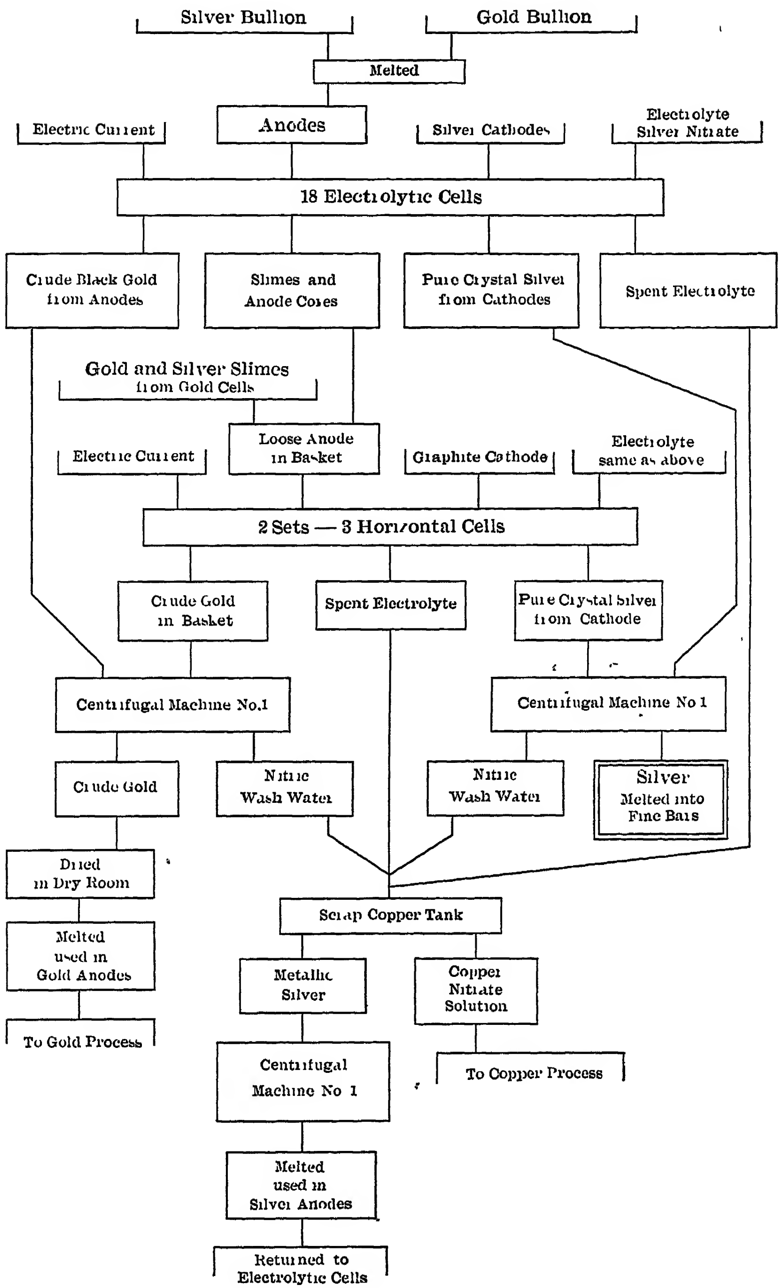


FIG. 1.—DIAGRAM OF THE SILVER PROCESS.

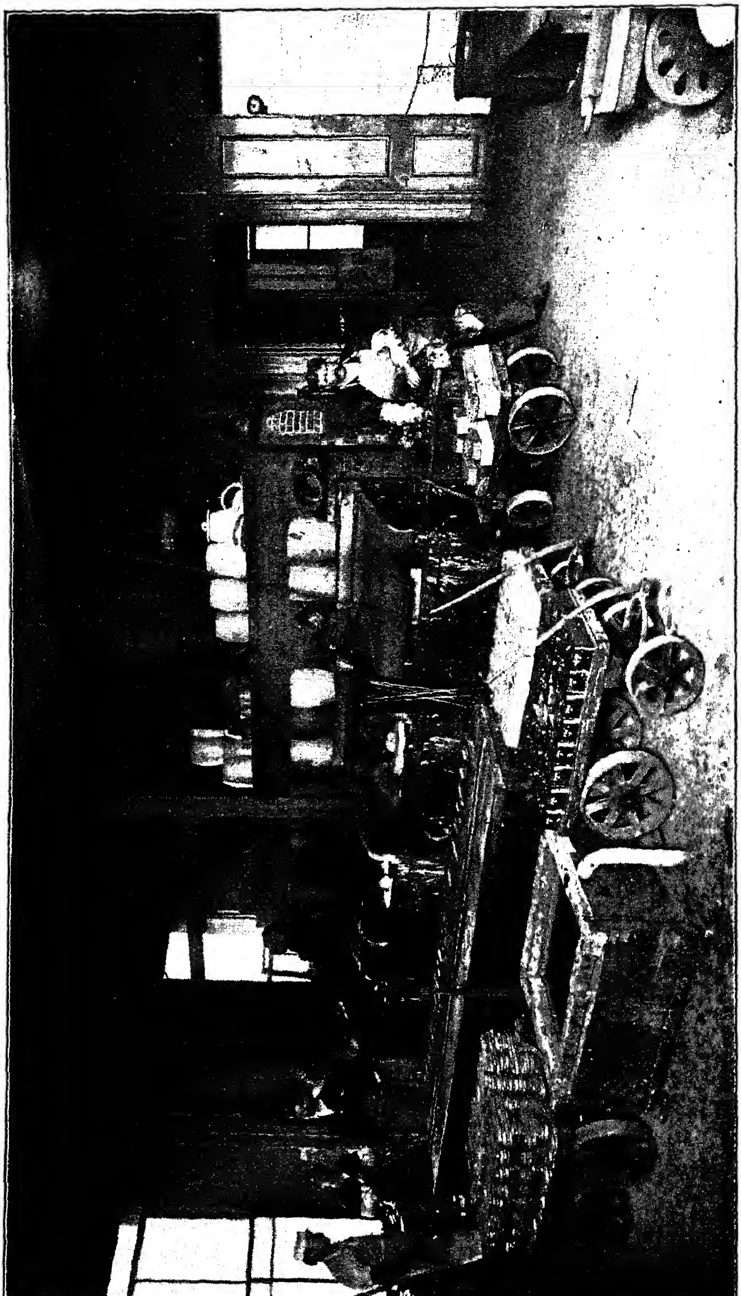


FIG. 3.—VIEW OF MELTING-ROOM.

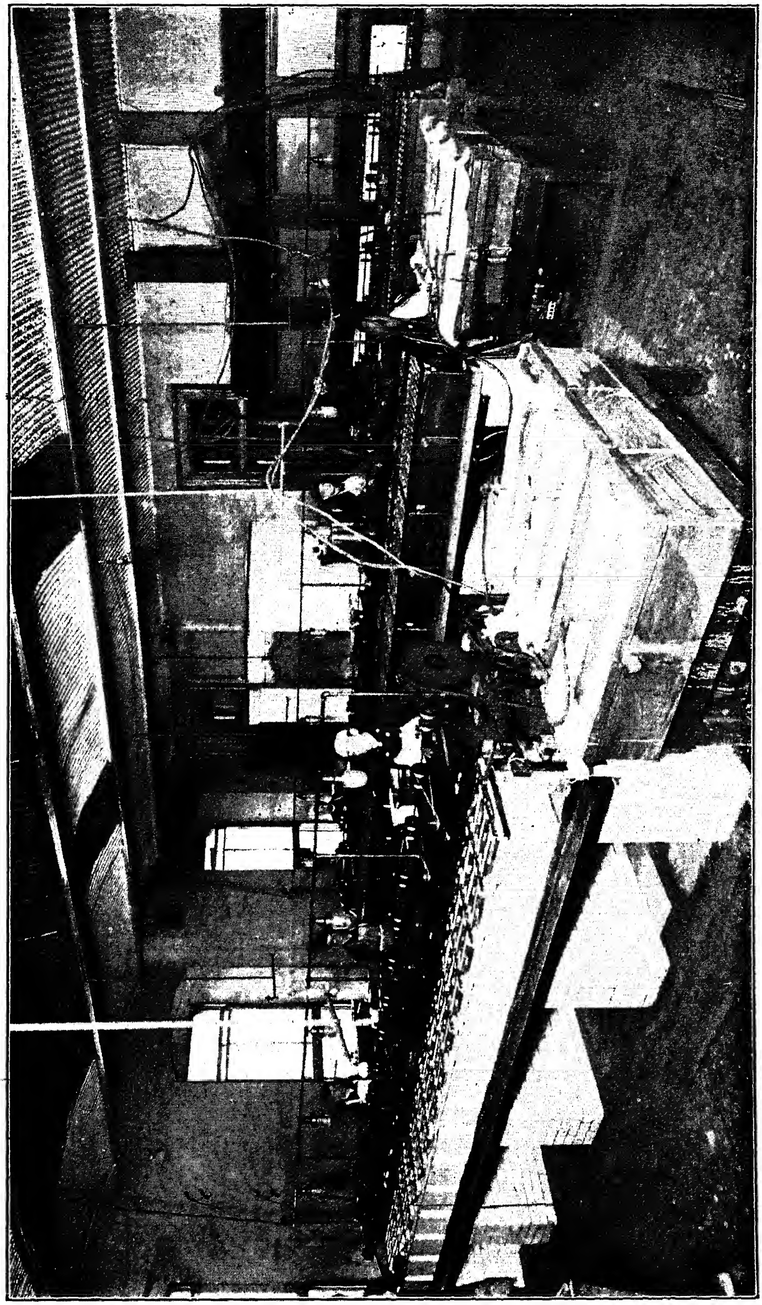


Fig. 6.—View of Cell-Room.

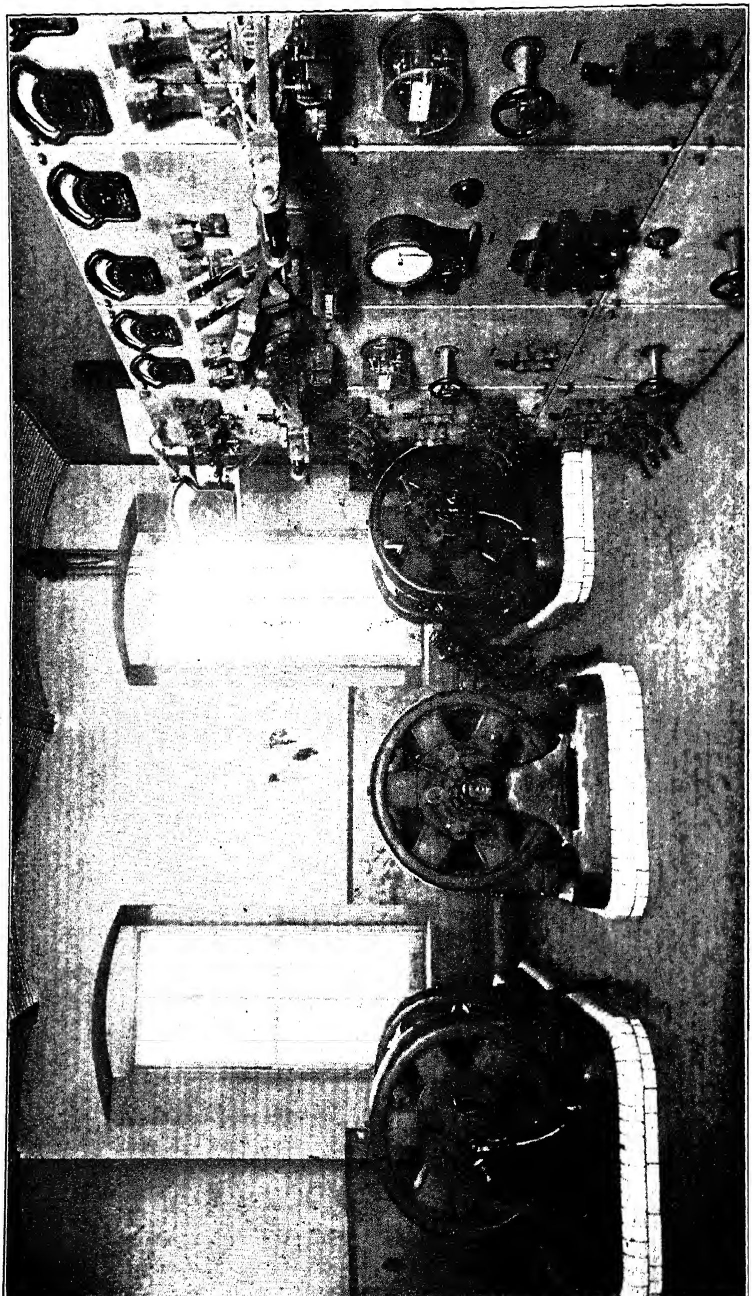


Fig. 8.—View of Generator-Room.

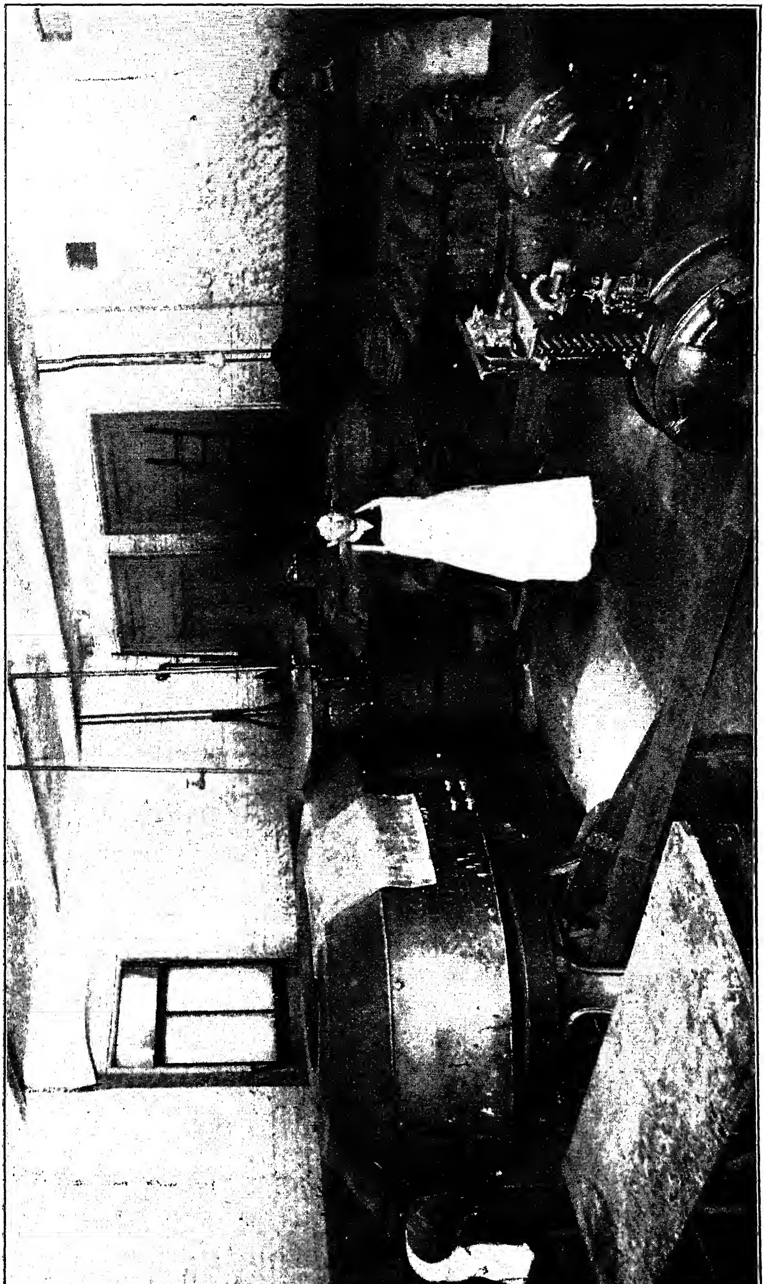


FIG. 9.—VIEW OF WASH-ROOM.

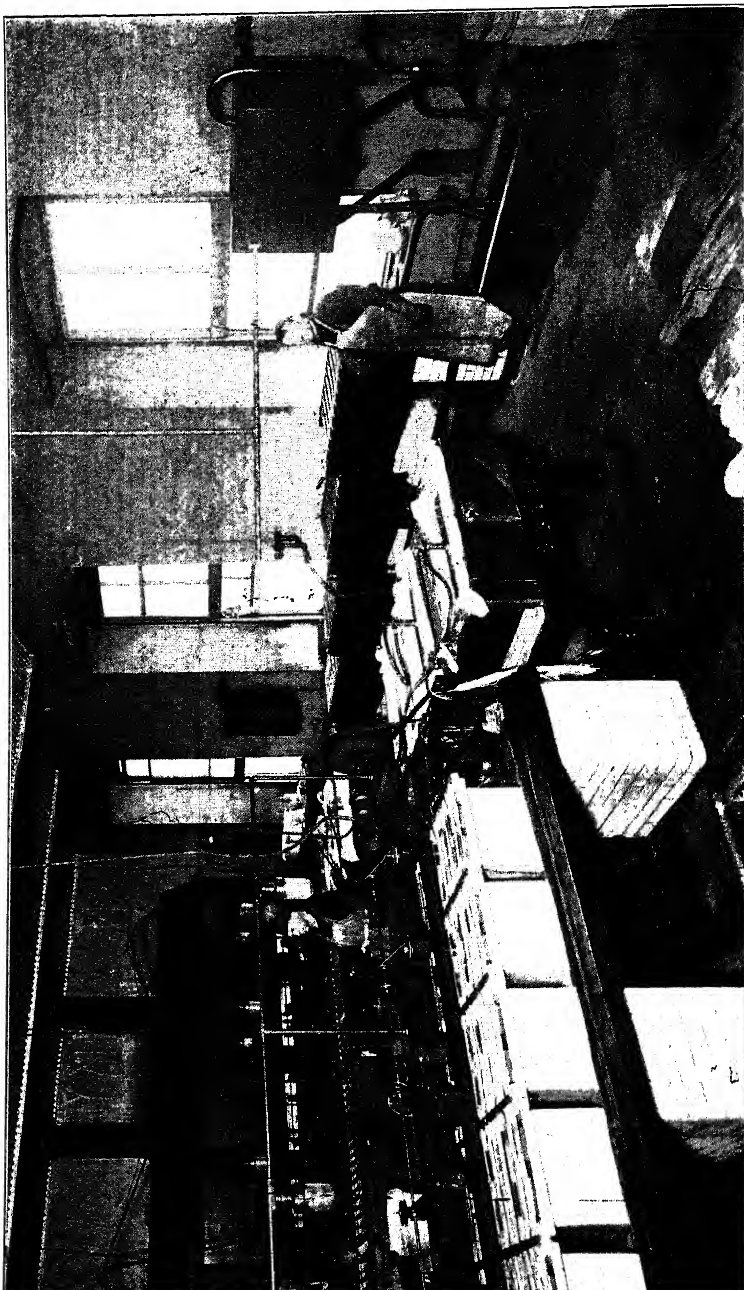


FIG. 14.—END OF CELL-ROOM.

A second product of this process consists of the slime that accumulates in the bottom of the cells. This contains black gold that has dropped from the anodes, as they dissolved, and also crystalline silver that failed to stick to the cathodes. This slime is transferred to the horizontal cells for re-treatment.

(In some plants, the anodes are incased in cloth bags, and the black gold is caught before it can drop to the bottom, and is melted for gold anodes, without further treatment.)

The operation in the horizontal silver-cells is the same in principle as in the vertical, but the mechanical details are different. There are two independent sets of the horizontal cells, each having three cells in series. These show at the right-hand end of the first and second benches in Fig. 6. The anodes consist of the cores of the silver anodes from the vertical cells, the slime from the bottom of the vertical silver-cells, and the silver reduced from the silver chloride slime from the gold-cells. These materials are contained in a wooden basket or tray. The current is led into this mass by a "candle," made of equal parts of gold and silver, the lower end of which is buried in the material. The cathodes consist of graphite plates on the bottom of the cells. The crystalline metallic silver is deposited on these cathodes, and is removed at intervals with a long-handled dipper of hard rubber. The electrolyte is the same as that of the vertical silver-cells. The current, about 50 amperes, passes through the three cells in series. This gives a current-density of 14.3 amperes per square foot of cathode surface, and requires a potential of 5 volts per cell, or a total of 15 volts.

The baskets are made of maple, and all the joints are dove-tailed, so that there is no metal in their construction. The bottoms are made with slats, and the baskets are painted all over with biturine solution. They are considerably smaller than the cells, so that the deposited silver can be scraped and gathered from the cathodes through the space between a basket and the side of its cell.

The material to be treated is retained on five layers of 7-oz. duck placed in each basket, and the edges are brought up on all sides above the top of the basket. This cloth shows as a white frill around the tops of these cells in Figs. 6 and 14. The baskets are suspended in the electrolyte by cleats resting on the tops of the cells.

The material left in the basket, after all the silver has been dissolved, is crude or black gold, and is transferred to centrifugal machine No. 1 and washed. It is then dried in the dry-room, melted, and used with other metal to make gold anodes for the gold process.

The spent electrolyte from both the vertical and the horizontal cells contains silver nitrate and the soluble nitrates of the base metals that were in the original bullion. These solutions and the nitric wash-waters from the centrifugal machine are passed over scrap-copper suspended in wooden tanks, which precipitates the silver and leaves the base nitrates in solution. These tanks are in the wash-room, as shown in Fig. 9.

The precipitated silver is washed and dried in centrifugal machine No. 1, and then is melted and cast into bars. These are added to melts of low-grade gold and made into silver anodes for the vertical silver-cells. At times, this precipitated silver has been dissolved in nitric acid to make silver nitrate for the electrolyte, but it is often impure, and a better electrolyte is obtained by dissolving pure silver; hence the practice is not common.

The solution containing the base nitrates is treated as described under the head of Copper-Refining.

II. Gold-Refining.

The process-tree, Fig. 10, gives an outline of the process of gold-refining, and shows the sequence of events in a graphic form.

1. *The Apparatus.*

A. The Anodes, of the same size as the silver ones shown in Fig. 4, are made from high grade gold-bullion and crude gold-products from both the gold and the silver refining processes. They carry about 90 per cent. of gold, and it is desirable that the silver-content be limited to about 7 per cent., since a greater amount interferes with the operations. Copper is less objectionable than silver. The metal for the anodes is melted in the furnaces shown in Figs. 2 and 3. The anodes, hung by C-shaped hooks of pure gold from the conductors running across the top of the cells, are immersed 7.5 in. in the electrolyte.

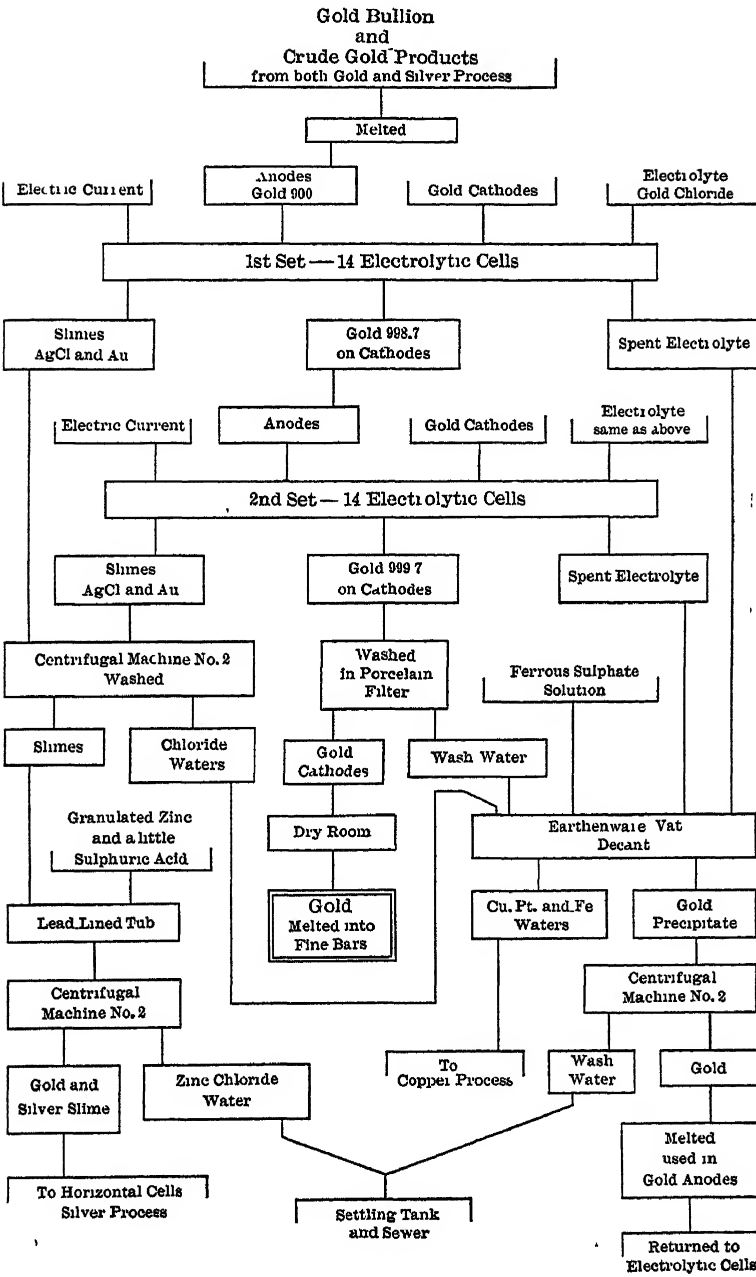


FIG. 10.—DIAGRAM OF THE GOLD PROCESS.

B. The Cathodes, strips of pure gold 4 in. wide by 0.012 in. thick (No. 28 B. & S. gauge), weigh about 4.5 oz. They are bent over at the top, so that they can be hooked over the conductors crossing the top of the cells. They are immersed to a depth of 6 in., and are allowed to remain in the cells until they weigh about 160 oz., when they are removed and used as the anodes for the second set of cells. By this re-deposition the fineness of the final product is raised to about 999.7.

The gold is deposited on the cathodes so tightly that stripping is impracticable, and when the final cathodes have been formed, the deposit with its original cathode sheet is all melted down together. Hence, the original strips have to be made of pure gold in order to maintain the quality of the product.

C. The Electrolyte is a trichloride solution, carrying in the first set of cells 70 g. of gold per liter, and from 10 to 12 per cent. of free hydrochloric acid, and in the second set, only 60 g. of gold per liter, but with the same amount of acid.

During the operation, the electrolyte decomposes and drops particles of metallic gold, which collect in the slimes. This lowers the strength of the solution in gold, and when it gets below 4 per cent. of gold, the deposit on the cathode is soft and tends to crumble. To prevent this, the bath is tested daily to determine its strength in gold, and if found to be low, is restored to the desired standard by the addition of strong solution.

The test of the electrolyte for gold is made with ferrous ammonium sulphate. A solution of this salt is made up of such strength that 1 cc. of it will precipitate 27.5 g. of gold. Then, to a liter of electrolyte is added 3.5 cc. of $Fe(NH_4)_2(SO_4)_2$ solution, which is capable of precipitating 96.25 g. of gold—more than the bath is likely to contain. The excess of the ferrous salt is then determined by titrating with potassium permanganate, using a solution such that 1 cc. of $K_2Mn_2O_8$ will oxidize 1 cc. of $Fe(NH_4)_2(SO_4)_2$. On dropping the permanganate into the solution, its purple color is destroyed as long as any of the ferrous salt remains, but when the latter is completely oxidized, an additional drop will retain its color, indicating the end of the reaction.

After a week, the electrolyte becomes spent and takes on a dirty dark-green color, due to the accumulation of copper-salts in the solution. When it reaches this condition, the gold-

deposit on the cathodes is soft, and the electrolyte has to be changed.

The gold chloride for the electrolyte is made by dissolving gold-bullion in hydrochloric acid by the aid of an electric current. Anodes of gold 990 fine are hung in strong hydrochloric acid, in five cells slightly larger than those used for the gold-refining process, and the cathodes, also of gold, are hung in porous cups filled with strong hydrochloric acid. On passing a current of 500 amperes at 25 volts through the cells, the anodes are dissolved, giving a solution of gold trichloride in the cells; but, owing to the porous cups, there is no gold deposited on the cathodes. Since hydrochloric acid fumes are

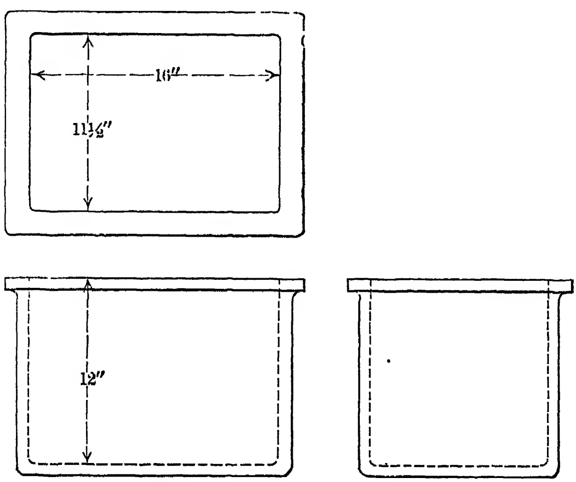


FIG. 11.—GOLD-CELL, OF WHITE ROYAL BERLIN PORCELAIN.

liberated in the process, it is performed under a glass-inclosed hood connected to a flue, shown in the right background in Fig. 6. The gold chloride solution obtained from these cells has a strength of from 375 to 500 g. of gold per liter.

D. The Cells are of white royal Berlin porcelain, and have the dimensions shown in Fig. 11. The electrolyte, like that in the silver-cells, already described, is kept in motion by one glass propeller in the center of each cell, revolved by a vertical glass rod.

The cells are placed in two rows, of 14 each, on a long bench. Those on one side form the first set, and those on the

other the second set, for re-treating the cathodes formed in the first.

The space between adjacent cells is covered with a porcelain strip about 1 by 3 in. in cross-section, clamped to the rim of the cells, and having a series of notches to receive the porcelain bars which support the conductors across the tops of the cells from which the electrodes are hung.

There are three rows of anodes and four rows of cathodes in each cell. The rows of anodes alternate with the rows of cathodes, and are $2\frac{5}{8}$ in. from center to center. There are two cathodes on each row, making eight cathodes per cell, and there are three anodes on each of two rows, but only two on the center row, making eight anodes per cell. The center anode is omitted to give room for the circulating propeller. The drive for the propellers is similar to that for the silver-cells. The arrangement of these parts is shown in Fig. 6, where the gold-cells (white) occupy the left foreground.

To the copper bus-bars, which are bolted to the top of the porcelain strips between the cells, are screwed the ends of the conductors that extend across the cells. These conductors are gold strips bent into an inverted trough shape, and fit the top of the porcelain cross-bars. The electrodes hang from these conductors.

E. The Current, a direct one of 15 volts potential, passes through the 14 cells of each set in series, as shown in Fig. 12, requiring nearly 1 volt per cell. The total amount of current is 180 amperes. There are eight cathodes in each cell in parallel, each having an immersed area of 4 x 6 in. = 24 sq. in. Four of the cathodes have both sides available for the reception of deposits and four have only one side available, thus making 12 cathode-surfaces of 24 sq. in. each, or a total of 2 sq. ft. The current being 180 amperes, the current-density is 90 amperes per square foot of cathode-surface.

F. Centrifugal Machine No. 2 is identical with No. 1, described under the silver process; but this one is used exclusively for gold-products and material charged with chloride waters, which would precipitate silver chloride if it came in contact with solutions of silver-salts. A different filter-bag is used for each kind of material. This machine is located in the wash-room (Fig. 9).

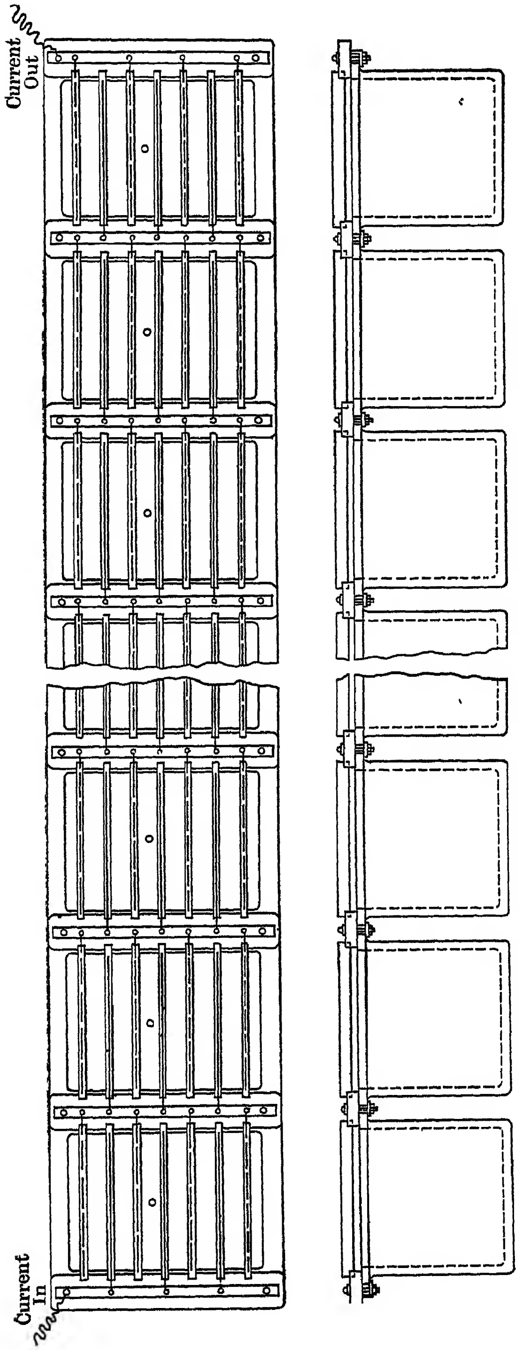


FIG. 12.—DIAGRAM OF PATH OF CURRENT THROUGH THE GOLD-CELLS.

G. The Drying-Room is of brick, has an iron door, is heated with steam and is built into one corner of the cell-room. It is about 5 by 6 ft. It shows in the central background of Fig. 6. It is used to dry fine gold cathodes, and other gold-products, before charging them into the melting-pots.

H. Vats and Tubs.—The vats used for the precipitation of the gold from the spent electrolyte are made of brown earthenware and stand on platform-trucks, for convenience in moving them about. They are 2 by 4 ft. in area, and 2 ft. deep.

The tub used for the reduction of the silver chloride to metallic silver, by means of zinc and sulphuric acid, is made of wood, and lined with lead. It is 2 by 4 ft., and 2 ft. deep, and mounted on a truck, similar to the earthenware ones.

2. *Operations and Products.*

Briefly, the anodes are dissolved in the electrolyte, and refined gold is deposited on the cathodes. All the metals in the anodes, including those of the platinum group, go into solution, except the silver and some lead. The last two form chlorides and drop to the bottom of the cells as the anodes dissolve. About 10 per cent. of the anodes is left as undissolved tops, and has to be remelted.

It is desirable that the anodes should not carry more than about 7 per cent. of silver. When more than this amount is present, the coating of silver chloride that forms on the anodes is thick enough to retard the dissolving action. When the anodes contain less than about 7 per cent. of silver, they can be treated in a single set of cells, and the gold-deposit on the cathodes will be considerably over 999 fine. But when more than 7 per cent. is present, so much silver chloride is formed at the anodes that, in dropping off, some of it is caught by the circulating currents, and carried mechanically to the cathodes, where it clings to the rough surface of the gold-deposit and lowers its fineness to less than 999. When handling such anodes high in silver, it has been found advisable to deposit the gold on the cathodes of one set of cells, and then transfer these cathodes, after washing them, to a second set of cells, where they are used as anodes and the gold is redeposited almost pure.

The gold anodes are made exclusively from the gold from the silver-cells, which assays about 875 thousandths gold, from 100 to 125 thousandths silver, and a small amount of base metals. This gives, in the first cells, cathodes about 998.7 fine, which, on being re-treated in the second set of cells, produce gold about 999.7 fine. It has generally been considered necessary to boil the crude gold from the silver-cells with concentrated sulphuric acid before casting it into anodes for the gold-cells, in order to reduce the silver to less that 7 per cent. The desire to do away with this acid treatment, and still produce a high grade of gold-deposit, led to the experiment of redepositing the first gold cathodes.

The same amount of current at the same voltage is used in both sets of cells. The electrolyte in the first set carries 70 g., that of the second set 60 g. of gold per liter. With the exception of this difference in the strength of the electrolyte, the operation in both sets of cells is identical.

The gold cathodes from the second set of cells are carefully washed in a porcelain filter, dried in the dry-room, melted and cast into fine bars about 1,000 oz. in weight, which may be sold as "mint bars," or alloyed with copper and made into coins.

The copper in the anodes goes into solution in the electrolyte; and as long as the proper amount of gold is maintained in the solution, it does no harm until the amount reaches about 4 per cent., when the gold begins to deposit soft and fall from the cathode. Then the electrolyte has to be changed.

The metals of the platinum group also dissolve in the electrolyte; and while they occur in such small quantities in the bullion that they can hardly be detected, the quantity accumulated in the solution by the dissolving of many anodes is quite appreciable, and is recovered as described later, under Copper-Refining.

The silver in the anodes forms at the anodes insoluble silver chloride, a part of which, in the first set of cells, is removed at intervals by taking out the anodes and brushing and jarring off the silver chloride into an earthenware jar. Most of the silver chloride, however, drops to the bottom of the cells.

The slime in the bottom of the cells also contains metallic gold, which comes from the decomposition of the electrolyte,

and does not deposit on the cathodes. This decomposition of the electrolyte seems to be due to the displacement of its gold by the copper dissolved from the anodes. In the first set of cells, with anodes containing 10 per cent. of silver, the slimes are about 600 thousandths gold and 300 thousandths silver, and in the second set, with anodes almost free from silver, they are 960 thousandths gold and only 40 thousandths silver.

The slimes from the bottom of the cells, and the silver chloride that has been removed from the anodes, are washed free from soluble chlorides in centrifugal machine No. 2, using hot water in order to carry off the lead chloride, and are treated in a lead-lined tub with granulated zinc, which precipitates the silver in a metallic condition, the zinc becoming zinc chloride. The granulated zinc is stirred into the mass of silver chloride and a little sulphuric acid is added to start the reaction. At first, the wet slime is a gelatinous mass characteristic of silver chloride, but as the reaction progresses it becomes more and more gritty. The mixture is tested towards the end of the process for the presence of silver chloride, and when there is no longer any present, sufficient sulphuric acid is added to dissolve any zinc that remains.

The test for silver chloride is made by treating a sample of the slime with ammonium hydrate, and then adding a few drops of hydrochloric acid to the clear solution. If there should be any silver chloride present, it would be dissolved by the ammonia, and would re-precipitate on adding the hydrochloric acid.

The granular silver with its gold-content, after being washed in centrifugal machine No. 2, to remove all soluble salts, is transferred to the anode-basket of the horizontal cells of the silver process for the recovery of the silver; and the gold is afterwards obtained from the basket-residue.

The wash-waters from the slimes and from the gold cathodes, together with the spent electrolyte from both sets of cells, are placed in earthenware vats, and a concentrated solution of ferrous sulphate is added to the liquid. This precipitates the gold, which is allowed to settle by long standing. The liquor, which still contains platinum-, copper-, and iron-salts, is decanted, and sent to the scrap-iron tank for further treatment

as described later under the head of Copper-Refining. The gold that remains after decantation is washed and dried in centrifugal machine No. 2, melted with low-grade bullion and cast into anodes, in which form it re-enters the process and is re-treated.

III. Copper-Refining.

This process is used at the San Francisco Mint to work up the copper occurring as base metal in the bullion, and to recover the copper used to precipitate the silver from the various wash-waters. It is similar to the commercial process of copper-refining; but it is of special interest here, because the metals of the platinum group, taken into solution in the previous operations, have now accumulated in sufficient quantities to be recovered. Fig. 13 gives a diagram of the process.

The wash-waters and spent electrolyte from all parts of the refinery, from which the gold and silver have been recovered, are sent to the scrap-iron tank, and there deposit their copper, lead, and any precious metals, including those of the platinum group, that have escaped from the previous operations. This tank is in the wash-room (see Fig. 9).

The sludge of cement-copper from this tank is washed and drained in wooden tubs with filter bottoms, whence it is transferred to other filter-tubs and allowed to air-dry, and then is melted down and cast into anodes for refining.

The copper anodes contain lead derived from the silver-bullion, metals of the platinum group derived from the gold-bullion, and small amounts of gold and silver. They are 5 by 14 in. by $\frac{5}{8}$ in. thick, and are immersed 13 in. in the electrolyte.

The cathodes are started on sheets of lead 3.75 by 15 in., and when both sides have been coated with a copper-deposit of sufficient strength, the copper is stripped off the lead and returned to the cells. This does away with the repeated melting and rolling of sheet-copper cathodes, similar to those of the precious metals used in the gold and silver processes. The cathodes are immersed 11 in. in the electrolyte and receive deposits on both sides. When completed, these cathodes are washed free of the electrolyte, dried, and added to melts of coin-metal, without previous melting into bars.

The cells are lead-lined wooden boxes, 3 by 1.5 ft. by 1.5 ft.

deep. Each cell contains 23 anodes and 24 cathodes, hanging in alternate rows, 2 in. apart from center to center.

The electrolyte is copper sulphate and contains 3 per cent. of copper as sulphate, and from 3 to 4 per cent. of free sulphuric acid. The cells are placed in a series of steps, so that the

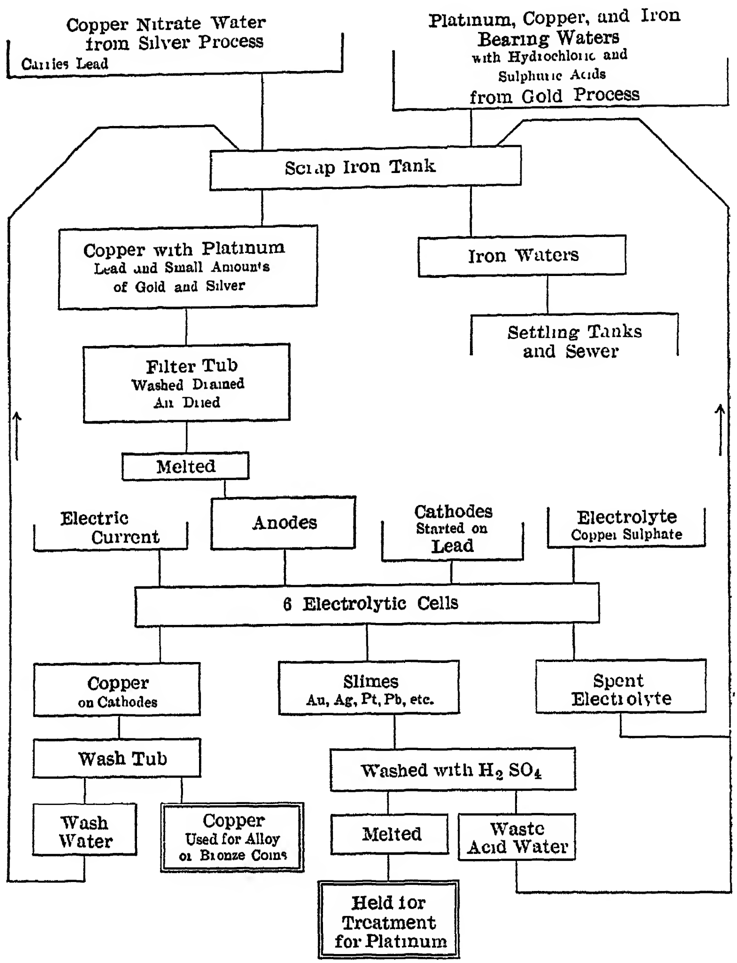


FIG. 13.—DIAGRAM OF THE COPPER PROCESS

electrolyte flows through them by gravity. A steam-ejector lifts the electrolyte from the sump at the lower end and returns it to the head-tank, from which it again flows through the cells. These tanks are shown against the wall in Fig. 14.

The current used is direct and has a density of 10 amperes per square foot of cathode-surface, and a potential of 3.6 volts, which is equal to 0.6 volt per cell.

The gold, silver, and metals of the platinum group are insoluble in the sulphate electrolyte, and drop to the bottom of the cells as slimes when the anodes are dissolved. These slimes are collected, washed with dilute sulphuric acid, dried, and melted into bars. These bars are stored until sufficient have accumulated, when they are treated for the separation of the various precious metals, especially those of the platinum group, that they contain.

IV. General Remarks on Mint Processes.

The Treasury Department maintains five refineries for the treatment of the gold- and silver-bullion deposited at the various mints and assay offices. The original installation in each case was the nitric acid process of refining. This was succeeded some 30 years ago by the sulphuric acid process, which in turn is now being displaced by the electrolytic process.

The electrolytic process was installed in the Philadelphia Mint in 1902, in the Denver Mint in 1906, and in the San Francisco Mint in 1908. It will be used in the New York Assay Office upon the completion of the new building; and the refinery of the New Orleans Mint, where the amount of work is comparatively small, will then be the only government refinery using the sulphuric acid process.

The mints and assay offices accept bullion carrying more than 200 thousandths precious metals. The refining-charges run from 1 cent an ounce on good silver-bullion, up to 8 cents an ounce on bullion carrying 800 thousandths base. The charges on ordinary gold-bullion average 4 cents per ounce. On account of these high charges on very base bullion, most of it is sent to private refineries, where the facilities for handling this grade of material are better, and the refining-charges are consequently less than at the mints.

In the silver process at the San Francisco Mint, the initial treatment of the bullion is in vertical cells. These are a modification, devised in the Philadelphia Mint, of the Moebius cells. The scraps from the vertical cells are re-treated in the hori-

zontal cells, which are a modification of the Thom cells. Both types of cells have their advantages and disadvantages.

For refineries where the silver-bullion is the product of cupel-furnaces, and carries less than from 50 to 60 thousandths gold, and not more than from 10 to 20 thousandths base metal, there is no question as to the superiority of the horizontal process.

In mint-work the case is different. The bullion carries from 100 to 150 thousandths base and from 300 to 400 thousandths gold; the base requires an excess of acid to put it in solution, and the large amount of gold necessitates current for parting, in addition to that needed to dissolve the silver. The presence of the excess acid and of the heavy currents tends to destroy the filter-cloths quickly.

The gold process used at all the mints is the invention of Dr. Emil Wohlwill, of Hamburg, Germany, and was the outcome of experiments to separate platinum from gold. It was introduced by him into several refineries in Europe, and was first installed in this country in the Philadelphia Mint; but, so far as I know, no private refinery in this country is using it.

The electrolytic process of gold-refining possesses three advantages that are important in mint-work. First, it produces purer gold than the old processes. The elimination of the last trace of silver from the gold removes the brittleness from the ingots used for coinage, so that they roll and press much better than alloys of the same fineness in gold, but made of slightly impure gold. Second, the process permits the saving of all the platinum metals without serious inconvenience. Third, the operations do not give off, as did former processes, great quantities of acid fumes, such as used to cause frequent complaints from the people living in the vicinity of the mints, which were all located in cities.

The electrolytic process of gold-refining has three disadvantages as compared with the sulphuric acid process. First, it is more expensive. Second, more care and intelligence are required to conduct it. Third, the losses are liable to be greater on account of having gold in solution in the electrolyte.

In mint-work, the advantages more than offset the disadvantages; but in commercial work, the advantages mentioned are of less importance, and the large amount of precious metal in-

vested in the process, with the resulting loss of interest, would be almost prohibitory of its use. This feature in not so important to the government, as the metal so tied up may be considered as part of the gold reserve, and is accounted for at the time of annual settlements.

V. References.

Electrolytic refining of gold, silver, and platinum is treated in the following articles:

D. K. Tuttle, *Electro-Chemical Industry*, vol. i., p. 157 (1903).

Emil Wohlwill, *Electro-Chemical Industry*, vol. ii., pp. 221 and 261 (1904).

Robert L. Whitehead, *Electro-Chemical Industry*, vol. vi., pp. 355 and 408 (1908).

Melting-operations of various kinds at the San Francisco Mint are described by Harold French in *The Pacific Miner* for December, 1909, and for January, 1910.

VI. Acknowledgment.

As already mentioned, I collected the notes from which this paper is prepared in December, 1909, and I wish to acknowledge the courtesies which were extended to me by E. R. Leach, melter and refiner, in showing all parts of the process, and in answering numerous questions. Mr. Leach has also furnished the photographs and lent his aid by valued criticism, both of the text and of the illustrations.

Phosphorus in Coking-Coal.

BY CHARLES CATLETT, STAUNTON, VA.

(San Francisco Meeting, October, 1911)

WHILE the occurrence of phosphorus in coking-coal has assumed less importance with the development of the open-hearth method of steel-making, it may not be without interest to note the form in which phosphorus exists in one particular coal-seam.

In the examination of what is known as the Big seam, which outcrops a few miles west of Columbiana, Ala., my attention was called to the distribution through the coal, in the form of minute veins and particles, of a resinous-looking substance. A small amount of this was selected, and was provisionally identified as evansite ($Al_6 P_2 O_{14}$. 18 Aq).

Subsequently, through the courtesy of Dr. J. Sharshall Grasty, of the Geological Department of the University of Virginia, I was able to secure an additional amount of material, which was purified down to about 0.3 g. This was examined by Prof. John J. Porter, of the University of Cincinnati, and gave the following partial analysis:

	Per Cent.
Loss on ignition,	37.43
Phosphoric anhydride,	10.33
Alumina,	36.33

There was also a trace of silica, and quite a considerable quantity of lime and magnesia.

Professor Porter was led to think that the material was not pure, but a mixture of several of the phosphates of aluminum carrying lime and magnesia. The material available did not permit of the convenient determination of the other ingredients.

One form in which phosphorus occurs in coal is evidently as a hydrated phosphate of aluminum; and any coal which shows to the eye the occurrence of a light-colored resinous-looking material should be looked on with suspicion as being high in phosphorus.

DISCUSSIONS.

Sampling Anode-Copper, with Special Reference to Silver-Content.

Discussion of the paper of William Wraith, *Trans.*, xli., 318 to 323.

EDWARD KELLER, Perth Amboy, N. J. (communication to the Secretary*) :—Mr. Wraith has done a real service to the art of sampling argentiferous copper by his extensive comparison of the two different methods of sampling—shotting and drilling—now in use. Experiments on so large a scale carry conviction to the business world, and demonstrate how great a degree of accuracy has been attained in a procedure on which very large financial transactions depend for a basis.

To Mr. Wraith's conclusion, that his own method of sampling the molten and homogeneous furnace-charge, by means of batting the running stream of metal at regular intervals, as it issues from the furnace, and thus obtaining "shots" for assay, is quite invariable in its results, I can readily subscribe. He has tested the drill-method with the regular 99-hole template, used at the refinery; and also in one variation—that of drilling one anode along two diagonals with eight holes in each. These tests gave results concordant with those of the shotting-method.

The somewhat restricted limits of these drill-method tests, and Mr. Wraith's remark that "a disagreement between the smelter and the refinery" led to his investigation, might seem to warrant the inference that there was something wrong in the drill-method of the refinery; in other words, that that method may be unreliable. Yet the Anaconda anode, which is the copper under consideration, is almost an ideal plate for drill-sampling. Its upper and lower surfaces are comparatively smooth; it has no blisters; it is thin and has large horizontal dimensions; and a drill-hole through the entire thickness in any part of the anode, except a narrow zone along the edge, should yield a sample fairly representing the whole.[1]

* Received Jan. 6, 1911.

[1] See my paper, The Distribution of the Precious Metals and Impurities in Copper, etc., *Trans.*, xxvii., 106 to 123 (1897).

In order to test this proposition, 99 anodes were drilled in four different ways:

1. From the top-surface, in each anode one hole through the 99-hole template; the holes being located in continuous order.

2. From the bottom-surface, in the same manner.

3. From the bottom-surface, one hole in the center of each anode.

4. From the bottom surface, one hole 1.5 in. from the edge, midway between lug-end and bottom-end of the anode, this location being chosen as lying in the uncertain edge-zone, and, therefore, likely to show the greatest deviation from the average silver-content obtained by means of the 99-hole template. The assay-results from the four samples are given in Table I.

TABLE I.—*Assays of Drill-Samples from Anaconda Anodes.*

Sample from	Silver. Oz. per Ton.
Top surface, through template,	80.838
Bottom surface, through template,	80.906
Bottom surface, center,	80.763
Bottom surface, edge,	81.052

These figures are the averages of 12 determinations for each sample, thus reducing to a minimum the errors of assaying. The differences are insignificant, and permit the conclusion that, no matter how we drill these anodes, within the confines of the template, the resulting samples are always well within practical, and permissible, limits of variation. This conclusion has been corroborated by other tests.

In connection with this discussion of the sampling, a few words should be said regarding its relation to the accuracy of the subsequent silver-assays.

The shot-sample has one inherent advantage over the drill-sample. It is homogeneous when it represents only one furnace-charge, which is itself homogeneous after the operations of refining; while the drill-sample from a single anode may be quite heterogeneous as to silver-content, since, in drilling through its thickness, that content changes from point to point, by reason of differentiation during cooling and solidification. This was at one time a serious objection to the drill-sample; but it is now overcome by very fine grinding. The samples from any of the Eastern copper-refineries pass through screens of from 16 to 20 meshes per linear inch.

TABLE II.—*Anaconda Anode-Copper.*

Average of Duplicate Silver-Assays; Ounces per Ton.

Drillings					Shot				
Sample	Assayer. A.	Assayer. B.	Differences.		Sample	Assayer A.	Assayer B.	Differences.	
1	83.17	82.87	0.30		21	79.00	79.01		0.01
2	82.92	83.77		0.85	22	80.09	79.92	0.17	
3	84.18	83.76	0.42		23	81.84	81.79	0.05	
4	82.72	82.64	0.08		24	84.05	84.33		0.28
5	84.42	84.46		0.04	25	86.93	86.82	0.11	
6	82.57	82.53	0.04		26	88.68	88.80		0.12
7	86.44	86.80		0.36	27	83.28	82.84	0.44	
8	85.21	85.39		0.18	28	80.90	81.16		0.26
9	87.33	87.31	0.02		29	82.62	82.25	0.37	
10	82.08	81.96	0.12		30	82.83	83.10		0.27
11	82.96	83.06		0.10	31	74.07	73.75	0.32	
12	82.78	82.75	0.03		32	79.22	79.18	0.04	
13	83.10	83.11		0.01	33	79.34	79.28	0.06	
14	85.65	85.69		0.04	34	82.12	82.09	0.03	
15	83.43	83.55		0.12	35	79.10	79.41		0.31
16	83.27	83.43		0.16	36	80.33	80.47		0.14
17	82.57	82.81		0.24	37	79.28	79.45		0.17
18	85.78	86.03		0.25	38	80.42	80.27	0.15	
19	83.09	83.46		0.37	39	74.15	74.39		0.24
20	84.88	85.10		0.22	40	77.83	77.81	0.02	
Average,	83.9275	84.024	0.0965		Average,	80.804	80.806	0.002	

Table II. shows the results of averages of duplicate assays (in this manner they are usually reported) by two men, A and B, on two sets of 20 samples each—20 drill-samples and 20 shot-samples. The differences in both series are very similar, indicating that they are due to manipulatory errors in assaying, rather than to the heterogeneity of either set of the samples. The results demonstrate also that if an accuracy within 0.1 oz. of silver be desired for the purpose of comparing sample-content and sampling-methods, from 10 to 20 assays per sample should be made; each set of assays to be carried out under like conditions in the furnace, etc.

Long ago, I pointed out the dangers of shot-sampling by ladle,[2] but found them not one-sided. Whether the resulting sample will be too high in silver when the copper is permitted partly to solidify, depends upon whether the pure copper or its impurities, inclusive of silver, freeze first. If the copper above its freezing-point be subsaturated, the pure copper will freeze first; the impurities will concentrate in the later-freezing portions, and the shot-sample will be too high in silver, as well as in all the other contained impurities. Nearly all converter-coppers belong in this category. If, on the other hand, the

[2] *Loc. cit.*

copper above its freezing-point be saturated with the same elements, the latter, upon cooling, will freeze first; the later-freezing portions will be purer copper, and the shot-sample will be too low in silver, etc. In this category are found practically all black coppers produced in blast-furnaces; generally speaking, such copper-materials as contain less than 97 per cent. of pure metal.

A Method of Calculating Sinking-Funds, and a Table of Values for Ordinary Periods and Rates of Interest.

Discussion of the paper of John B. Dilworth, *Trans.*, xli., 533 to 535.

JOHN LANGTON, New York, N. Y. (communication to the Secretary*):—In Mr. Firmstone's discussion (*Trans.*, xli., 912) the formula he gives for the periodical payment—his equation (2)—may be simplified in form. The expression given by Mr. Firmstone is:

$$Q = \frac{S(1+r)^n}{1 + \dfrac{(1+r)^n - (1+r)}{r}}$$

$$\text{this} = \frac{S(1+r)^n}{\dfrac{r + (1+r)^n - (1+r)}{r}}$$

$$\text{or} = Sr\left(\frac{(1+r)^n}{(1+r)^n - 1}\right) \quad . \quad . \quad . \quad . \quad (I)$$

which is a simpler form to use in calculations.

Where S = principal to be extinguished.
Q = periodical payment or charge for sinking-fund.
r = rate of interest per period (*i. e.*, interest on \$1 per period).
n = number of payments or periods.

Kent's *Mechanical Engineer's Pocket Book*, under the heading *Annuities*, equation No. 5, gives:

"The annuity which \$1 will purchase for any number of years n is $\dfrac{r}{1 - \dfrac{1}{(1 \quad r)^n}}$"

* Received July 16, 1911.

The periodical payment Q is an annuity for n years (or periods) purchased by the sum S. Hence by Kent's formula:

$$Q = S\left(\frac{r}{1 - \frac{1}{(1+r)^n}}\right)$$

$$= Sr\left(\frac{(1+r)^n}{(1+r)^n - 1}\right) \quad . \quad . \quad \text{same as } (I)$$

Having recently had occasion to calculate fixed periodical charges for a sinking-fund, on the basis of the legal rule for partial payments stated by Mr. Firmstone, I desired to verify the formula given by Kent before applying it. I found that it could be proved very simply, as follows:

In any one of the n equal periodical payments, let

$x =$ that portion of the payment which equals the interest due;

$y =$ that remaining portion of the payment applied to the principal.

Then $S = y_1 + y_2 + \quad . \quad . \quad . \quad . \quad + y_{n-1} + y_n \quad . \quad . \quad (A)$

and $Q = x_1 + y_1 = x_2 + y_2 = \; . \; = x_n + y_n \; . \quad . \quad . \quad (B)$

Also $x_1 = Sr$

$x_2 = (S - y_1)\, r$

$x_3 = \{S - (y_1 + y_2)\}\, r$

$x_n = \{S - (y_1 + y_2 + \quad . \quad . \quad . \quad . \quad + y_{n-1})\}\, r.$

Therefore, substituting in (B)

$Q = y_1 + Sr \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (C)$

and $Q = y_2 + (S - y_1)\, r$

and $Q = y_3 + \{S - (y_1 + y_2)\}\, r$

and $Q = y_n + \{S - (y_1 + y_2 + \quad . \quad . \quad . \quad . \quad + y_{n-1})\}\, r$

therefore $y_1 + Sr = y_2 + (S - y_1)\, r \quad . \quad . \quad .$ or, $y_2 = y_1 (1 + r)$

and $y_2 + (S - y_1)\, r = y_3 + \{S - (y_1 + y_2)\}\, r$ or, $y_3 = y_2 (1 + r)$

and similarly $y_n = y_{n-1} (1 + r)$

Whence it appears that (A) is a series in geometrical progression with a common ratio $= (1 + r)$.

Therefore its sum, $S = y_1 \dfrac{(1+r)^n - 1}{(1+r) - 1}$

$$\text{whence} \qquad y_1 = \frac{Sr}{(1+r)^n - 1}$$

and, substituting this value of y_1 in (C)

$$Q = Sr\left(\frac{1}{(1+r)^n - 1} + 1\right)$$

$$= Sr\left(\frac{(1+r)^n}{(1+r)^n - 1}\right) \quad . \quad . \quad \text{same as } (I)$$

In applying this formula care must be taken to give r its correct numerical value. For instance, if the rate of interest is 5 per cent. per period, r in the formula is 0.05 and not 5.

Mine-Survey Notes.

Discussion of the paper of George W. Riter, *Trans.*, xli., 790 to 796.

E. R. Rice, Wickenburg, Ariz. (communication to the Secretary*):—While this paper is primarily intended as a discussion of Mr. Riter's, I think it will be best to indicate my criticism by describing my own field-methods. It has been my experience that, for ordinary work, the regular transit-book is to be preferred to the card-system for recording notes. This is specially true when computations in the field are required, for it is then necessary to have at hand the total latitudes, departures, and elevations. Moreover, the transit-book is easier to carry and manipulate in wet or cramped places, and is not as liable to damage as the loose leaf or card.

I use a regular transit-book in the field, and then copy my notes, sketches, etc., in an office-book, entering also the latitudes, departures, and other reductions. These values are then copied into the field-book, securing a duplicate record, in case either book should be lost or mislaid.

In the field-book, the notes are entered on the left-hand, and the sketches and remarks on the right-hand page. The next two pages are left blank for the latitudes, departures, bearings, reduced distances, etc. Of course, the notes for the different parts of the mine are entered in the field-book in the order in which they are surveyed; but in the office-book they are entered systematically.

It has been my experience that the system of keeping notes used by a surveyor, is particularly adapted to the needs and temperament of the individual. Otherwise, he would not be

* Received May 24, 1911.

using it. Yet we can all generally learn something from each other; and I describe the system of notes which I employ, in the hope that some one may find something useful in it.

Everything we employ in engineering, whether method or machine, involves two necessary requisites: it must be accurate; and it must be practically "fool-proof." We can all heartily agree with Mr. Riter when he says, "So seldom does a surveyor have a chance to check underground surveys by making a closure, that he is compelled to rely on the precision of each step of his work for the accuracy of the final result."

In surveying, there are three sources of error to be guarded against, namely: (1) errors in reading the vernier and tape; (2) errors in recording the readings obtained; and (3) instrumental errors. To guard against the first and second, it is necessary to take duplicate readings and measurements on both fore- and back-sights, and to throw all of the reading of the tape on the transit-man, who, by reason of his superior intelligence and training, is better qualified for this work. The third source of error is eliminated by the proper manipulation of the instrument.

The notes should be in such a form that all the duplicate and doubled readings can be recorded without confusion. They should also permit the entry of side-notes necessary for the making of a correct map. They should be simple and easily understood, and should necessitate the recording of as few items as possible. Tables I. and II. give the notes for two courses, as taken from my field-book. The notes are identical, the difference in them being in the position at which the height of the instrument and the height of the point are recorded. Of these two forms, I use the second exclusively, since it permits all the notes to be put on the left-hand page of the note-book, leaving the right-hand page free for remarks and sketches. The second set calls for an entry in every space except one. The notes here given are for two set-ups of the transit: one at station "D," and one at station "E." Stations "C," "D," and "E," are stations in an incline shaft and station "500" is the first station in the 500-ft. level. The instrument is first set up at station "D," the station occupied by the transit being recorded in the Station column. The back-sight is taken on the point "C," and the back-sight station is entered in the Point column, on the same horizontal line as that occupied by

the instrument-station. The fore-sight is taken on the point "E," and the fore-sight station is entered in the Point column, on the line below the back-sight station.

The height of the instrument—*i. e.*, the vertical distance from the horizontal axis of the instrument to the point under or over which the instrument is set—is entered in the Height of Instrument column on the same line as the instrument-station. The "height of point" is the vertical distance, above or below the line of sight, of the point sighted. This is entered in the Height of Point column, on the same horizontal lines as the station to which it refers. In reading horizontal angles, I always set the vernier at zero on the back-sight, and turn the angle to the fore-sight, reading the plate in azimuth, up to 360°. The reading of the vernier on the back-sight is recorded in the Plate column, on the same line as the back-sight station, and the reading of the vernier on the fore-sight is recorded also in the Plate column, on the same line as the fore-sight station. The difference between the two readings is the difference in azimuth between the fore- and back-sights.

The plate is always read in azimuth, because the vernier is then always read in one direction, and there is no necessity of recording whether the angle was read to the right or the left, as is the case when deflection-angles are employed.

After the plate has been read on the fore-sight station, the lower motion is unclamped, the telescope is plunged, and the back-sight is bisected with the cross-wires, by means of the lower slow-motion screw. The upper motion is then unclamped; the telescope is turned on to the fore-sight; the plate is again read; and the reading is recorded just below the first reading, and on the same line. If the instrument is in perfect adjustment, and both readings have been made and recorded correctly, the last reading will be just twice the first.

This doubling of the horizontal angle, with the telescope inverted, serves three purposes: (1) By taking one-half of the last reading as the true value of the angle, the horizontal angle can be more closely determined than if the one reading were made; (2) this method shows whether the plate has been read, and the reading recorded, correctly; and (3) by taking one-half of the last plate-reading as the true angle, all errors due to the lack of adjustment of the line of collimation and

the horizontal axis are eliminated. The latter consideration is very important when the sights are inclined.

The vertical angles, or the readings of the vertical circle, are recorded in the Vertical Circle column. The vertical angle of the line of sight to the back-sight is recorded on the same line as the back-sight station, and that to the fore-sight on the same line as the fore-sight station.

By reading the vertical angles, and measuring the distances, of both the fore- and back-sights, and taking the mean of the horizontal and vertical distances obtained as the true distances, index-errors of the vertical circle, and errors due to the lack of adjustment of the bubble attached to the telescope, are eliminated. I notice that Mr. Riter reads the vernier of his vertical circle both direct and reversed, to guard against mistakes in reading the vertical circle.

After recording the vertical angle, I set my vertical circle so that the vernier reads 17 min. more or less than the recorded reading, and then see if one of the stadia-hairs cuts the point sighted at. The distances measured are entered in the Distance column, on the same line as the points to which the distances are measured. If a distance is measured horizontally, then the reading of the vertical circle will be zero.

In measuring distances, I always measure from the axis of the instrument to the point sighted at. The vertical circle reading then gives the inclination of the tape from the horizontal. I always make the chain-man hold the zero of the tape at the point sighted, while I read the tape at the axis of the instrument. Under this procedure, if any mistakes are made, I make them; and I am not always bothering as to whether the chain-man read the tape correctly or not.

By taking measurements on both the fore- and back-sights, I have an absolute check on myself. This is a refinement in ordinary work; but where the survey is important, it is absolutely necessary. It does not take long to make the extra measurement and reading, and by so doing, and taking the mean of the results obtained from the fore- and back-sight measurements, systematic instrumental errors are kept from accumulating.

The side-notes go on the line below that occupied by the fore-sight to which the side-notes refer. If more than one point is sighted from the set-up, the other fore-sights go on the line below the side-notes of the preceding fore-sight.

In the notes here given, the transit is set up under "D," the height of instrument being —4.02 ft., and the back-sight is taken on the head of a plumb-bob suspended from "C." The height of the back-sight station above the line of sight—*i. e.*, above the head of the plumb-bob—is + 5.19 ft., and the slope-distance from the axis of the instrument to the head of the plumb-bob is 80.55 ft., and the vertical angle of the line of sight is + 30° 46′ 30″.

The vernier of the horizontal circle is set at zero on the back-sight, as indicated in the Plate column. The horizontal angle is turned to "E" and the plate read in azimuth, the reading being 179° 58′ 30″. The fore-sight is taken on the head of a plumb-bob suspended from "E," the height of which is + 5.36 ft. The vertical angle of the line of sight is — 35° 22′ 0″, and the distance from the axis of the instrument to the head of the plumb-bob is 109.94 feet.

The lower motion is then unclamped, the telescope is plunged, and the plumb-line at the back-sight is again sighted. The upper motion is then unclamped and the plumb-line at station "E" is again sighted. The plate is read in azimuth, and found to read 359° 57′ 0″. As this is twice the first reading, we are sure that our first angle is correct. The side-notes are then recorded. The distances from the instrument towards the fore-sight are recorded as whole numbers, and the distances from the line of sight to the walls are recorded as fractions, the numerator of the fraction being the distance from the line of sight to the left wall, and the denominator the distance to the right wall. Thus, at the instrument, the distance from the instrument to the left wall is 3.6 ft. and to the right 3.3 ft. At 20 ft. from the instrument, it is 3.2 ft. to the left wall and 4.1 ft. to the right; and so on.

If it is necessary for mapping to get the outlines of the floor and roof, the same scheme can be used—recording the distance from the line of sight to the roof as the numerator, and the distance to the floor as denominator.

A reduction of the above notes will show that from the data obtained at station "D," the horizontal distance between "D" and "E" is 89.65 ft., while the vertical distance is 62.29 ft. From the data obtained at station "E," the horizontal distance is 89.64 ft. and the vertical 62.3 ft. As these values agree with each other, and as one-half the doubled horizontal angle is

equal to the single angle at station "D," we are sure that our work is correct.

It is a great deal easier to compute the bearing of a line from its azimuth than in any other way. When the azimuth is known, its bearing can be determined mentally.

Rule: To find the azimuth of a line, add to the azimuth of the preceding line the horizontal angle and 180°. Thus: The bearing of the line C–D is N. 45° 2′ 30″ E.; hence its azimuth is 225° 2′ 30″. Therefore, to find the azimuth of the line D–E, we proceed as follows:

	225° 02′ 30″
	179° 58′ 30″
	180° 00′ 00″
	585° 01′ 00″
Less	360° 00′ 00″
	225° 01′ 00″

225° 01′ 00″ is the azimuth of the line D–E; hence its bearing is N. 45° –1′ –0″ E.

To get the bearing of the line E–500, we would proceed as follows:

	225° 01′ 0″
	76° 50′ 0″
	180° 00′ 0″
	481° 51′ 0″
Less	360° 0′ 0″
	121° 51′ 0″

121° 51′ 0″ is the azimuth of the line E–500; hence its bearing is N. 58° 09′ 00″ W.

For accurate work, it is essential that the instrument be perfectly level; and since the ordinary plate-levels are too sluggish, and generally not quite in adjustment, I level the transit for important work by means of the bubble attached to the telescope, after approximately leveling it by means of the plate-levels.

In computing the vertical and horizontal distances, as well as the latitudes and departures, I use a Gurdens traverse-table, and check the results by means of a slide-rule. The system of notes here given can be used with equal facility for either underground- or surface-work. In ordinary surface-work, the height of instrument, height of point, and the back-sight vertical circle, and D readings, as well as the side-notes, can be omitted.

TABLE I.—*Record of Field-Notes.*

Station.	Point	Height of Instrument.	Height of Point.	Plate.	Vertical Circle	D.
D	C	— 4.02	+ 5.19	0 0	+ 30° - 46′ - 30″	80 55
				179 - 58 - 30		
	E		+ 5.36	359 - 57 - 00	— 35 - 22 - 00	109.94
0 $\frac{3.6}{3.3}$	20 $\frac{3.2}{4.1}$	40 $\frac{4.0}{3.0}$	60 $\frac{3.6}{3.4}$	80 $\frac{3.5}{3.5}$ 10	0 $\frac{4.0}{3.0}$	
E	D	— 4.08	+ 6.18	0 0	+ 33 - 53 - 30	107.98
				76 - 50 - 00		
	500		+ 3.84	153 - 40 - 00	0 0	18.20
6 $\frac{\text{S. P.}}{8.0}$	7 $\frac{1.5}{6.5}$	10 $\frac{0.1}{5.5}$	18 $\frac{2.0}{2.4}$			

TABLE II.—*Alternative and Preferable Record of the Field-Notes Given in Table I.*

Station	Point.	Plate	Vertical Circle.	D.
— 4.02 D	+ 5.19 C	0 0	+ 30 - 46 - 30	80.55
	+ 5.36 E	179 - 58 - 30 359 - 57 - 00	— 35 - 22 - 00	109.94
0 $\frac{3.6}{3.3}$	20 $\frac{3.2}{4.1}$	40 $\frac{4.0}{3.0}$ 60 $\frac{3.6}{3.4}$	80 $\frac{3.5}{3.5}$	100 $\frac{4.0}{3.0}$
— 4 08 E	+ 6.18 D	0 0	+ 33 - 53 - 30	107.98
	+ 3.84 500	76 - 50 - 00 153 - 40 - 00	0 0	18.20
6 $\frac{\text{S P}}{8.0}$	7 $\frac{1.5}{6.5}$	10 $\frac{0.1}{5.5}$ 18	$\frac{2.0}{2.4}$	

The Agency of Manganese in the Superficial Alteration and Secondary Enrichment of Gold-Deposits in the United States.

Discussion of the paper of William H. Emmons, p. 3.

CHARLES R. KEYES, Des Moines, Ia. (communication to the Secretary*):—It is not in a spirit of criticism that I offer a supplemental suggestion or two on the subjects covered by this valuable and highly instructive memoir. There are two points which, in my opinion, should have received greater emphasis in Mr. Emmons's excellent paper. One is the fundamental rôle played by the chlorides under certain conditions in ore-formation. The other is the possible establishment of geographic relationships among the four phenomena of (1) excessive chloridic content of mine-waters; (2) the abundance of chloridic compounds of the precious metals; (3) the presence of manganese oxides; and (4) the diminishing importance, in ore-genesis, of the metallic sulphates.

Although Mr. Emmons's notes refer to gold alone, it may be pertinently asked whether the same principles do not hold good for silver and copper also, since these metals form, together with gold, a distinct and well-known chemical group. That the reactions involved apply equally well to the other two metals mentioned, is shown by a number of recent observations and discussions. Chloridic ores of silver are, as I have lately endeavored to show, mainly worked in arid or desert regions only; and the great deposits of disseminated copper-ores are similarly characteristic of such regions, in which both classes doubtless owe their formation to the abundance of saline materials derived from desert dusts, and to the plentiful and almost universal presence of manganese oxide. Under these climatic conditions, silver is somewhat more abundantly deposited than copper and gold, because its chloride is so much less soluble. During volcanic emanations, the metallic chlorides perform at all times distinct functions; when these are

* Received Nov. 11, 1910.

over, the metals pass into other combinations. Thus, in dry climates, the genetic rôle of the chlorides of the metals is strictly analogous to that of the sulphates of the metals under conditions of moist climate.

The possible solution of gold in cold ground-waters has given more concern to the scientist than to the miner, because the former has had to find adequate proofs of this contention. Yet a deep-rooted notion has long prevailed, among Western placer-miners especially, that, after a time, worked-over gravels renew their gold-content, and become pay-ground again. I was long ago led to believe that, in many instances at least, the miner was right. It may be that throughout arid regions placers are often developed not with, but long after, the deposition of the gravels. Many of the gold-bearing "cement-beds," which are old consolidated layers of gravel, or coarse rock-waste, appear in many ways to support this view. The famous, long-worked placers of the Ortiz mountains, in Santa Fe county, New Mexico; the Animas Peak placers, of Sierra county, New Mexico; and the recently opened Altar deposits in central Sonora, Old Mexico, are to be especially considered in this connection. One has only to conceive the gravel-bed as a once-porous layer, favorably situated for interstitial ore-deposition, in order to recognize all the conditions for the formation of a disseminated ore-body. The actual physical conditions are identical with those under which the porphyry coppers occur. By the adoption of new methods for handling such deposits, they might be made, perhaps, as attractive as the disseminated coppers now are.

From the strictly industrial side, the zone of secondary sulphide-enrichment is, of course, very important, because it supplies to-day, and is likely to supply for a long time to come, the bulk of the ores mined. Being now generally regarded as mainly the result of downward-percolating meteoric waters, it gives solid support to certain aspects of ore-genesis, which those who are committed to a strictly igneo-genetic theory are very loath to admit. As I have lately pointed out, the zone of secondary sulphide-enrichment, or "the bonanza-zone," as I prefer to call it, is to be regarded not as a mere local phenomenon, but as one of world-wide extent. The vadose zone of the ores may thus be considered much in the same way as is the

regolith, everywhere underlain by a sub-zone of bonanza character, which in some places is sufficiently well-developed to be mined, but in others only feebly represented by ore-materials.

Too much reliance cannot be placed upon either the importance or the distinctness of so-called metallo-genetic epochs in the earth's history. As these are emphasized by Mr. Lindgren, they in fact correspond closely to the "critical periods" of Professor LeConte, and refer chiefly to North America only. In other parts of the world, as is well known, and as Sir Archibald Geikie has recently urged, such "volcanic revolutions" take place in all intermediate epochs. It seems probable that ever since pre-Cambrian times the epochs of volcanic activity in general have not been more distinctly marked than they were during the latest geologic times, or than they are to-day. Moreover, in view of the well-established fact that igneogenetic metalliferous deposits do not always, or perhaps not even in the majority of cases, accompany volcanic manifestations, the questions arise: When do they? And when do they not? Generalization from wider observations than we now have may show that they are formed mainly, and perhaps only, when laccolitic conditions involving the contact-metamorphism of rocks prevail. The attempt to recognize distinct metallogenetic epochs probably obscures rather than illuminates the practical problems involved.

Miners often make important empirical observations which are subsequently confirmed by scientific generalizations. The genetic association of the precious metals with manganese oxide is an illustration. I well remember with what astonishment, upon first becoming acquainted with the arid mining-regions, I beheld the unerring promptness with which, from the presence of "wad," miners inferred the existence of rich ores in districts entirely new to them. I soon found that under this homely title the notion very generally prevailed among them that the black oxide of manganese was one of the surest indicators of values. My first test in prospecting along these lines yielded me ores carrying 10 per cent. of copper and $36 per ton in gold; and I stood convinced. Since that time I have always given the closest attention to even the merest prospects in which the presence of manganese oxide was conspicuous.

Although manganese oxide has been long known as an important aid in the solution of gold in the vadose zone, the subject appears never to have received the attention it merits. Dr. Pearce's results, published in 1893, have been generally overlooked, because they appeared in a channel little known to mining-men. The published notes of the wide observations of Mr. Rickard and of the careful and extensive experiments of Dr. Don on the same subject seem also strangely to have escaped deserved notice. Of similar import are the somewhat later observations of Professor Lehner on lead oxide as promoting the solution of gold in the presence of salt. The work of these investigators, as well as the more recent labors of Messrs. Stokes, Brokaw, and Emmons, goes to corroborate strongly the empiric rules long ago laid down by the prospector.

It seems more than a coincidence that essentially the same geographic boundaries should serve for mine-waters containing excessive amounts of common salt, for an abundant occurrence of chloridic ores, for a great prevalency of manganese oxide in the vadose zone, and for a notable deficiency of sulphate compounds of the metals. I recently ventured to suggest that, in the case of the horn-silvers at least, their prevalency as ores was due primarily to conditions imposed by desert climate; and still more recently I have called attention to the fact that all four features mentioned are characteristic of arid regions.

The mines named by Mr. Emmons, in which manganese oxide is abundant, are chiefly located in the deserts of the Great Basin. The vast Colorado Plateau of Arizona, the California Gulf basin, and the Mexican table-land would furnish even more convincing evidences of the abundance of this mineral and its intimate connection with the precious metals. Like relationships appear to prevail throughout the dry regions of South America, Australia, South Africa, and western Asia; so that Mr. Emmons's work has a much wider bearing than he has claimed for it.

Mine-Caves Under the City of Scranton.

Discussion of the paper of Eli T Conner, p. 246.

RUFUS J. FOSTER, Scranton, Pa. (communication to the Secretary*):—In answer to one of the inquiries made of Mr. Conner, and as a matter of historical record, I beg to say that the idea of supporting the strata over worked-out areas in coal-mines by flushing the openings full of culm originated and was put into actual practice by the late R. C. Luther, a former member of the Institute, at that time Chief Mining Engineer, and later General Manager, of the Philadelphia & Reading Coal & Iron Co.

In the early 80's the Philadelphia & Reading Coal & Iron Co. purchased from Messrs. Hecksher & Co. the Kohinoor colliery, situated in the western part of the borough of Shenandoah, Pa. On taking possession of the colliery the Philadelphia & Reading Coal & Iron Co. made a complete resurvey of the mine-workings, giving an absolutely correct map which showed not only the plan of the mine-workings, but also every improvement on the surface over the workings, and the marked features of the surface of the ground. Tidal elevations were shown at every survey-station in the mine and on the surface, so that, as is the case with all the maps of the company, contour-maps of either the surface or the bottom of any coal-seam, or geological cross-sections, could be constructed directly from the information on the mine-maps.

In a comparatively short time after the company took possession of the colliery, the workings which extended under the western portion of the borough of Shenandoah began to cause trouble and damage dwellings and other buildings in that section of the town. The lots in this section had been sold, in many cases, to individuals by the Gilbert & Shaefer Estate, owners of the coal-lands on which the colliery was located. In

* Received June 17, 1911.

selling these lots, while the mineral right was reserved, there was no clause in the deed, such as is inserted in nearly all the deeds in the city of Scranton, which released the operators of the mine from any damage to the surface or buildings thereon caused by mining coal. The Gilbert & Shaefer Estate repurchased all the lots possible, but on many of them substantial homes had been erected, and the natural increase in the value of the real estate, plus a possible little cupidity on the part of the owners, made it impractical to purchase all of them.

As the workings approached the Roman Catholic church and rectory, which was probably the most expensive property in that section of the town, Mr. Luther realized that if there was a material disturbance of the surface, not only would the operating company be liable for heavy damages on account of the intrinsic value of these buildings, but that there was a sentimental or reverential value to the structures that would have to be considered.

In 1886 he originated the idea of filling the workings with culm. With the very complete mine-map available it was an easy matter to construct a contour-map of the floor of the Mammoth seam, which, by the way, in this place ranged from 40 to 60 ft. thick normally, and which, owing to a peculiar geological formation, doubled back on itself, making a seam of from 80 to 120 ft. thick. In addition, several cross-sections were constructed showing the thickness and character of the strata between the surface and the top of the coal-seam. Bore-holes 8 in. in diameter were sunk with ordinary churn-drills at points so located as to secure the maximum flow of flushed culm in the mine. Pumps were installed to pump water from a convenient stream to the bore-holes, and scraper-lines were put in to convey culm from the large culm-piles to the holes. The culm was then flushed into the mine with the water, and it packed very solidly. As the chambers filled up occasional cross-cuts were driven through pillars into adjoining chambers so as to run the flushed culm into them.

At that time I was connected with the engineering department of the Philadelphia & Reading Coal & Iron Co., and was a member of the corps having this work in charge, and on several occasions I was in the mine and on top of the flushed culm where it ranged in thickness from 60 to 100 ft. This

culm was packed very solidly and compactly by the flushing; the water draining off and flowing to the sump near the foot of the shaft, where, with the ordinary mine-drainage, it was pumped to the surface and flowed into a neighboring stream, to be used over and over again.

After the desired area to be protected was filled there were large quantities of pillar-coal of superior quality which could be taken out, and in some instances short gangways were driven through the culm to reach the pillars. The driving of the gangways or headings through the culm was an easy matter, the fore-poling method of timbering being used. It is my impression that after the pillars were taken out the spaces they occupied were also filled with culm; but of this I cannot speak definitely, as shortly after that time I left the service of the company.

Note.—Since writing the above, I have learned that in the latter part of 1885 there was an extensive squeeze in the second and third levels of the Laurel Hill mine, of Messrs. A. Pardee & Co., at Hazelton, Pa. The squeeze was creeping slowly to the west and passed the special timbering as fast as it was put in place. Frank Pardee, then Assistant General Superintendent for A. Pardee & Co., suggested to his father, the late Ario Pardee, and his brother Calvin, then General Superintendent, the plan of flushing two breasts, between the slope and the squeeze, with culm, through bore-holes. The plan was carried out and the result was a complete success. This prior use of the system of flushing, on a comparatively small scale, shows that the credit for first using it belongs to Mr. Pardee. In justice to Mr. Luther, however, it must be recorded that he was unaware of Mr. Pardee's use of culm for the purpose of supporting the overlying strata when he made the plans for the later work at Kohinoor colliery.

Geology of the Cobalt District, Ontario, Canada.

Discussion of the paper of Reginald E. Hore, p. 480.

Cyril W. Knight, Toronto, Ont., Canada (communication to the Secretary*).—Mr. Hore's paper presents an interesting summary of our knowledge of this important mineral field; and is therefore acceptable as giving information which our *Transactions* did not already contain, although it presents little or nothing not made known by other writers before 1906, when Mr. Hore began work in the field as a student assistant. Contributions of this kind, however, should be made completely valuable by such references to previous work as will enable the reader to follow to their original sources the theories or conclusions stated by the author. It is quite natural, and even excusable, that a new observer should find novelty in his own observations—especially if he is not acquainted with the work of predecessors in the same field. Moreover, the independent repetition of an observation or conclusion has a distinct value as a confirmation, though it may have none as a discovery. But the publication of it, without reference to its predecessors, may place the author in an unfavorable light as either ignorant of the history of his subject, or willing to claim credit to which he is not entitled. Of the latter fault, I do not accuse Mr. Hore; but I cannot wholly acquit him of the former. Without disparaging his intelligent work, I must point out that his paper, as it stands, might easily give the impression that he claims originality for many statements which are by no means new, and that he emphasizes unduly his own publications.

Thus, his remark (p. 481) "I have described the silver-fields in a general way in my paper" (not yet published), should have been accompanied with reference to some of the numerous papers, already published, which describe these fields.

Again, on p. 486, he refers to four papers of his own on the Porcupine gold-area, but neither in the text nor in the appended

* Received May 29, 1911.

bibliography does he mention papers on this subject by other authors.

On the same page, he says, "In numerous instances I have found Huronian conglomerates which lie unconformably on granites and syenites referred to the Laurentian." Since, in numerous other instances, numerous other persons had made the same observation, Mr. Hore's use of the personal pronoun should have been guarded against misunderstanding. It looks as if he considered himself as the discoverer of this relationship.

With regard to the resemblance of the conglomerates to glacial material, Mr. Hore's single reference (p. 492) to a paper of his own, might be construed as showing ignorance or disregard of the discussion of this subject by other writers before he ever saw the Cobalt region.

In connection with the diabases (p. 492), with aplitic veins (p. 493) and with the origin of the ores (p. 497), Mr. Hore's foot-notes cite no publications except his own. It may be that in his own papers, thus cited, full credit is given to earlier investigators (Van Hise, among others) who published discussions of these subjects before Mr. Hore had published anything. But he should have remembered that any such acknowledgments of earlier work do not come before the readers of our *Transactions* unless they happen to be also readers of the other publications cited. I have no doubt he will heartily disclaim the interpretation of his statements to which he has unintentionally laid himself open.

Unfortunately, the bibliography appended to Mr. Hore's paper is incomplete in some important particulars. I think I am doing a service to both him and his readers by supplying the following items:

Porcupine Gold-Area.—A geologically colored map, with notes, was published by the Ontario Bureau of Mines, July, 1910. The notes give a fairly full description of the gold-veins and the geology in general.

Conglomerates Lying Unconformably on Granites.—There are numerous references to this relationship in the *Reports of the Canadian Geological Survey* and of the *Ontario Bureau of Mines*. References are made to a number of such occurrences in the Report of the Special (International) Committee for the Lake Superior Region, *Journal of Geology*, vol. xiii., 1905, pp. 89–104.

Glacial Origin of Conglomerate.—W. G. Miller refers to this in the *Report of the Ontario Bureau of Mines*, 1905, Part II., p. 41. A. P. Coleman has also

assumed a glacial origin for the conglomerate. *American Journal of Science*, vol. xxiii., March, 1907; and *Journal of Geology*, vol. xvi., 1908, p. 149.

Aplitic Veins.—These are described in the *Reports of the Ontario Bureau of Mines*, vol. xvi., Part II., pp. 65, 124–125.

Origin of Cobalt-Silver Ores.—This is discussed in the *Report of the Ontario Bureau of Mines* of 1905, Part II., p. 7, and later *Reports;* also, by C. R. Van Hise in the *Journal of the Canadian Mining Institute*, vol. x., 1907, pp. 45–61.

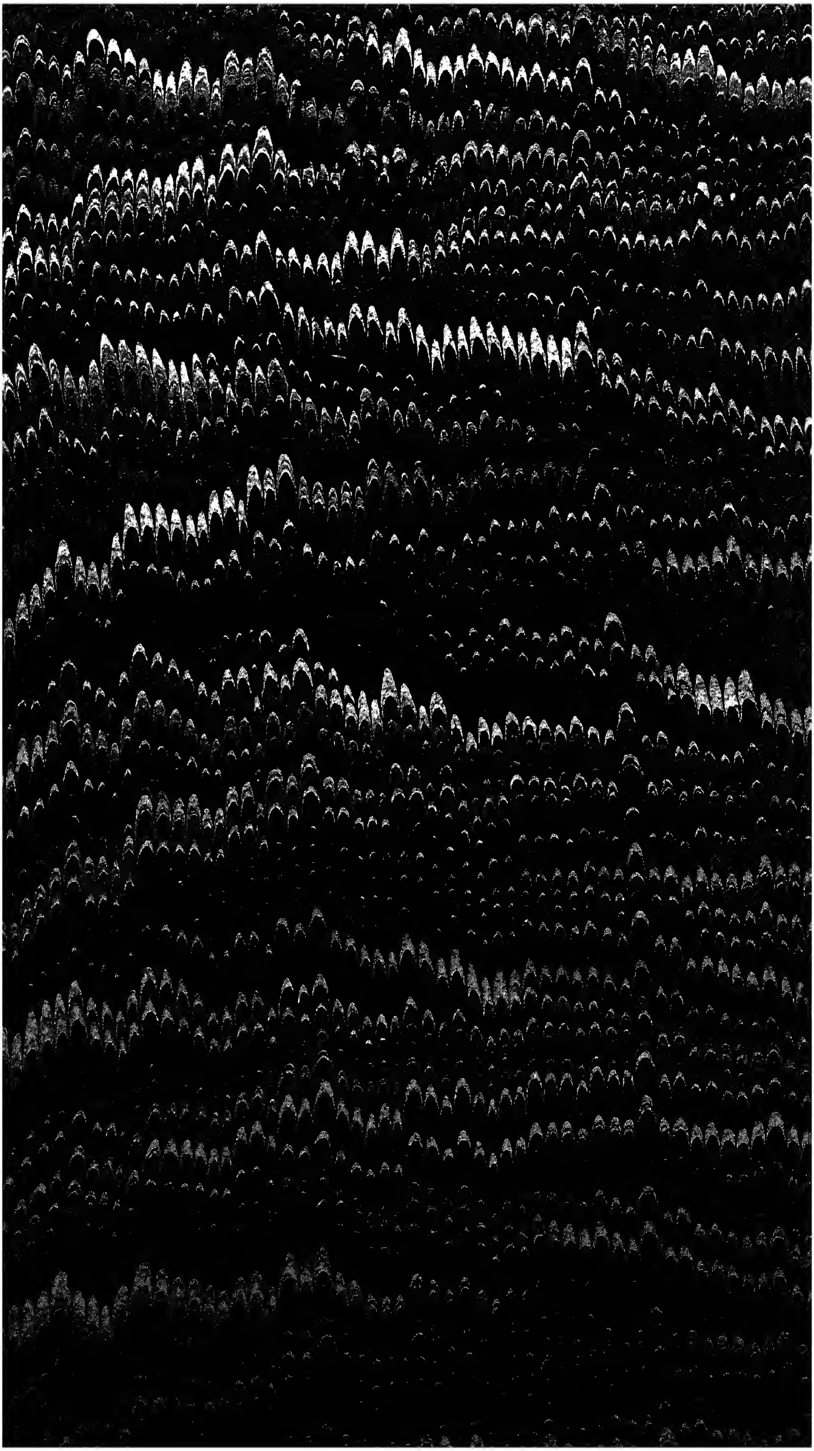
As to the character of the diabase and the geology of the silver-area in general, the literature is voluminous.

INDEX.

[NOTE.—In this Index the names of authors of papers are printed in small capitals, and the titles of papers in italics. References to papers expressly treating of the subject named are likewise in italics; and casual notices, giving but little information, are usually indicated by bracketed page-numbers. The titles of papers presented, but not printed in this volume, are followed by bracketed page-numbers only.]

ERRATA.

P. 552. Equation (3).
  For c $(P_2 - P_1)$, read c $(P_1 - P_b)$.
P. 739. Table of Extraction, 1909 column.
  Per cent. recovered by amalgamation. For 64, read 60
P. 740. Summary of Costs, 1908 column.
  Salaries and office. For .28, read .29.
  Miscellaneous. For .06, read .02.



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